NI 43-101 TECHNICAL REPORT FOR THE EVA COPPER PROJECT—FEASIBILITY STUDY UPDATE

NORTH WEST QUEENSLAND, AUSTRALIA

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1 EXECUTIVE SUMMARY

The Eva Copper Project (the Project) is 100% owned by Copper Mountain Mining Corporation (CMMC, or the Company) through a wholly-owned subsidiary Copper Mountain Mining Pty. Ltd. (CMMPL). The Project is in North West Queensland, approximately 76 kilometres (km) northwest of Cloncurry, and 194 km northeast of Mount Isa.

The Eva Copper Project is anticipated to mine 170 million tonnes (Mt) of ore and 381 Mt of waste from seven open pit deposits, with a minimum projected mine life of 15 years. The seven deposits in order of size are Little Eva, Blackard, Scanlan, Turkey Creek, Lady Clayre, Bedford, and Ivy Ann. Mineral Reserves will be mined using conventional earthmoving equipment, and will be hauled to a processing plant by way of haul roads from each pit. Waste material will be stacked in waste dumps adjacent to each pit, except for some material that will be used to construct the tailings storage facility (TSF) and bund walls around the open pits.

The processing plant will process 11.4 million tonnes per annum (Mt/a), operating at 31,200 tonnes per day (t/d), through a conventional crushing, high pressure grinding rolls (HPGR), milling, gravity, and flotation plant, for the fifteen-year life-of-mine (LOM) duration.

Existing major infrastructure closely surrounding the Project site includes the Burke Developmental Road, located 8.5 km to the east of the Project, which connects Cloncurry with Normanton. A power transmission line installed by MMG Limited (MMG)'s Dugald River mine is located 11 km south of the Project. A water pipeline that runs from Lake Julius to the Ernest Henry Mine traverses the southern portion of the Project site. A residential area, known as the Mount Roseby Homestead, is located approximately 17.5 km to the south of the Project plant site. Current infrastructure located on the Project site itself is minor, and includes dirt tracks for exploration, water points, and fences.

Major infrastructure required to be developed for the Project includes:

- Processing plant, workshops, laboratory, administration, security, and training offices
- Seven open pit mines, pit dewatering, diversion channels, and bund walls
- Tailings Storage Facility
- An 11 km, 220 kV power transmission line from the Dugald River mine
- An employee accommodation village to house 300 personnel
- New intersection from the Burke Developmental Road, an 8.5 km-long site access road, and haul roads
- Water wells at Little Eva, Blackard, and approximately 2 km north of the Little Eva pit
- Telecommunications infrastructure.

1.1 Key Facts

Units of measurement used in this report conform to the metric system. All currency is United States dollars (US$) unless otherwise noted.

Table 1-1 shows a list of facts for the Eva Copper Project described in this NI 43-101 Technical Report.

Table 1-1: Eva Copper Project Summary

Mineral Resources and Mineral Reserves (sulphide only)		Tonnes (Mt)	Copper Grade (%)	Gold Grade (g/t)	Contained Metal	
					Copper (Mlb)	Gold (koz)
Total Mineral Resources	– Measured and Indicated	260.7	0.42	0.04	2,419	330
	– Inferred	46.3	0.42	0.04	415	51
Total Mineral Reserves	– Proven and Probable	171.0(1)	0.46	0.05	1,718	260

	Feasibility Study 2018	Feasibility Study Update 2020
Production Summary		
Project life (years)	12	15
Eva Copper Project strip ratio (waste:ore [w:o])	1.9:1	2.2:1
Average processing rate per annum (Mt/a)	9.687	11.388
Copper recoveries, % (sulphide/native copper ores)	93	87
Gold recovery, %	78	78
Copper concentrate grade (%)	25	28
Milled tonnes over LOM (Mt)	117	170(1)
Target grind (µm)	225	165
Recoverable copper, LOM (Mlb)	959	1,502
Recoverable gold, LOM (koz)	203	205
Average copper in copper concentrate, (Mlb/a)	76.7	99.2
Average gold in copper concentrate, (koz/a)	15.5	13.5
Costs		
Initial capital cost ($ million)	350	443(2)
Operating costs ($/t milled)	12.11	11.39
Operating (C1) cash cost per pound copper after credits ($)	1.74	1.44
LOM sustaining capital ($/lb)	0.11	0.02
Royalties ($/lb)	0.16	0.13
Project Economics		
LOM revenues after smelter charges ($ million)	2,851	4,140
Total LOM free cash flow ($ million)	556	1,091
After-tax NPV (8.0% discount rate) ($ million)	256	437
After-tax internal rate of return (IRR) (%)	28	29
Assumptions (Long Term)		
Copper price	3.08	3.04
Gold price	1,310	1,362
AU$ to US$ exchange rate	1.32	1.55

Note: (1)There is a 661 kt difference between the Mineral Reserve at 171.047 Mt and the LOM schedule at 170.386 Mt. The Mineral Reserves were computed in Maptek's Vulcan software, which uses proportional blocks to compute volumes. The LOM mining schedule was generated in Geovia's Mine Scheduler, using whole blocks to compute volumes.
(2)After preproduction revenue of $11.2 million.

1.2 Project Overview

Copper Mountain Mining Pty. Ltd. (CMMPL) is a wholly-owned subsidiary of Copper Mountain Mining Corporation (CMMC, or the Company). CMMPL is located in Queensland, Australia, and was formerly known as Altona Mining Limited (Altona). The Project is located approximately 76 km northwest of Cloncurry in North West Queensland, Australia, and has extensive exploration potential in the approximately 4,000 km^2 (379,000 hectare [ha]) mineralized land package.

CMMC commissioned Ausenco Limited (Ausenco)to redesign and redevelop the 2018 Feasibility Study process plant and associated site infrastructure, and to provide technical input into the preparation of this National Instrument (NI) 43-101-compliant Feasibility Level Technical Report. In addition, CMMC commissioned Klohn Crippen Berger (KCB) to redesign the 2018 Knight Piésold Ltd. (Knight Piésold) TSF and to provide input to water management, and Merit Consultants International (Merit), a division of Cementation Canada Inc., to develop the capital cost, construction management, and execution plan of the Project.

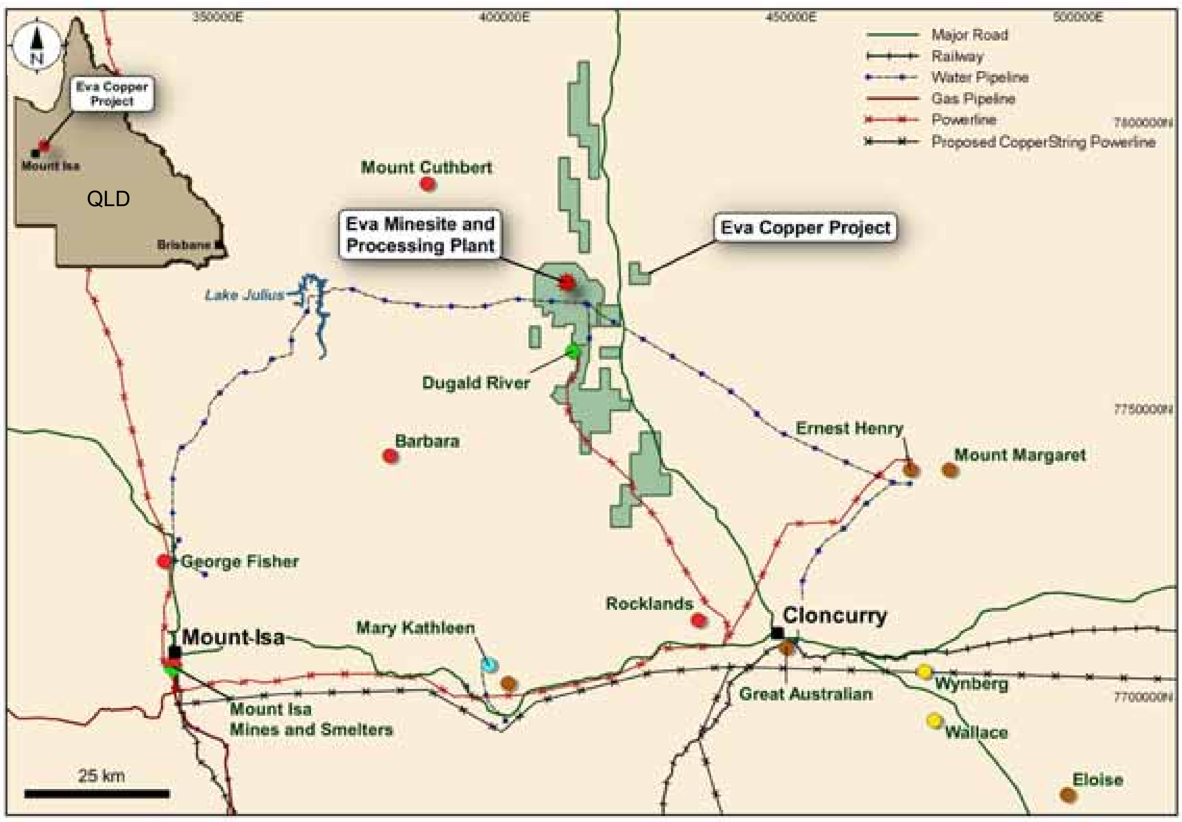

Figure 1-1: Eva Copper Project Location, Tenure, Plant, and Regional Infrastructure

The Project is proposed to be a large, open pit copper-gold mining operation with an associated gravity and flotation processing plant, similar to other operations in the Mount Isa and Cloncurry area. The Project comprises the large Little Eva open pit and six smaller satellite pits, which will deliver a sulphide and native copper ore mixture in a ratio of 75% to 25%, respectively, to a 11.4 Mt/a processing plant adjacent to the Little Eva and Turkey Creek pits.

The Little Eva deposit was the subject of a major drilling programme from 2010 to 2012, which consequently more than doubled the deposit's contained Mineral Resources. The enlarged Little Eva deposit was the focus of many feasibility studies, comprising a simple operation treating copper-gold sulphide ore. However, after the 2018 CMMC Feasibility Study, CMMC performed additional infill drilling on the Blackard deposit in 2019, and subsequently included the Blackard and Scanlan deposits in this updated Technical Report, thereby increasing the capital cost, but improving the Project reserves by 45% (from 117 Mt to 170 Mt), the Project NPV by 71% (from $256 million to $437 million), and the LOM recoverable copper by 57% (from 959 Mlb to 1,502 Mlb).

The process plant redesign was guided by extensive additional metallurgical testwork, and is in several ways similar to the Company's processing plant near Princeton, British Columbia, Canada, the New Afton processing plant near Kamloops in British Columbia, Canada, and the Ernest Henry processing plant in Queensland, 60 km distance from the Eva Copper Project.

It is estimated that over 28 years a total of $46.9 million has been expended on exploration, resource development, metallurgical and engineering studies, compensation payments, government fees, and charges by Altona's predecessor, Universal Resources Limited (Universal), Universal's partners, and by parties who held the Project prior to Universal. Altona spent approximately $21.0 million from February 2010 through March 2018, and CMMC has spent $4.8 million since taking ownership of CMMPL.

Responsible for specific report sections, the qualified persons (QPs) as defined under NI 43-101 (by virtue of their education, experience, and professional association, and their membership or good standing with appropriate professional institutions or associations) are as follows:

- Paul Staples, Mining and Metals VP and Global Practice Lead, Ausenco Limited (Ausenco)
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- David Johns, Senior Geotechnical Engineer, Klohn Crippen Berger (KCB)
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- Roland Bartsch, Vice President and Country Manager Australia, Copper Mountain Mining Pty. Ltd. (CMMPL)
- Richard Klue, Vice President Technical Services, Copper Mountain Mining Corp. (CMMC).

This report is based on a combination of inputs from Ausenco, CMMC, CMMPL, Merit, KCB, Knight Piésold, MBS Environmental (MBS), and Rockwater Hydrogeological Consultants (Rockwater).

1.3 Reliance on Other Experts

The QPs' opinions contained herein are based on public and private information provided by CMMC and others throughout the course of the study. The authors have carried out due diligence reviews of the information provided to them by CMMC and others for preparation of this report. The authors are satisfied that the information was accurate at the time of writing, and that the interpretations and opinions expressed are reasonable and are based on a current understanding of the mining and processing techniques and costs, economics, mineralization processes, and the host geological setting. The authors have made reasonable efforts to verify the accuracy of the data relied on for this report.

1.4 Property Description and Location

The Eva Copper Project is located 76 km northwest by road from Cloncurry, and 194 km northeast by road from Mount Isa, a regional mining centre. Access to the Project is via the sealed Burke Developmental Road from Cloncurry. This road passes 8.5 km to the east of the proposed processing plant site and the Little Eva and Turkey Creek pits. The site is also 11 km north of the major operating Dugald River zinc mine.

The Project is 100% owned by CMMC. The planned pits and Mineral Resources are within five granted Mining Leases (ML), except for the Ivy Ann pit, which is within the Exploration Permit for Minerals (EPM) 25760 (King). The MLs total an area of 143 km^2, and are situated across from two pastoral lease holdings and within one Native Title grant. There are two freehold lots granted in the late 1800s, and 100% owned by the Company, that lie within the MLs; the first sits over part of the Little Eva deposit, the second over part of the Longamundi deposit.

Necessary agreements are secured with the pastoral leaseholders and Native Title party (Kalkadoon People) that set out conduct and compensation terms for the planned mining activities to proceed. Additional third-party agreements and consents have been secured for the Project access road from the Burke Developmental Road. An application has been submitted to the Department of Natural Resources, Mines and Energy (DNRME) for the realignment of the mine access road proposed in the current design.

Numerous royalties apply to the Project. Royalties on minerals are payable annually to the Queensland State Government on an ad valorem basis, with various costs being permitted as a deduction from sales revenue. Copper and gold royalty rates vary between 2.5% and 5.0% of value, depending on average metal prices, as per Schedule 3 of the *Mineral Resources Regulation* of 2003. No state royalty on copper is applicable to the two freehold lots owned by the Company Several royalties also apply to the Project from purchase agreements and are payable to several parties variably across portions of the Project area. These apply to all of the deposits in the Project mine plan: a total 1.5% net smelter return (NSR) royalty is applicable to the Little Eva, Blackard, Scanlan, Turkey Creek, Bedford, and Lady Clayre deposits, and a 2% NSR royalty is applicable to the Ivy Ann deposit. Compensation for the effects of mining activities on the Native Title of the Kalkadoon People has been agreed upon.

In addition to the granted MLs, the key environmental and permitting consideration for a mining project in Queensland is the approved Environmental Authority (EA) from the Department of Environment and Science (DES), the administrating authority for the environmental management of the Project.

The Queensland Government introduced rehabilitation and Financial Assurance (FA) reforms subsequent to grant of the current EA and previous Feasibility Study that included the *Mineral and Energy Resources (Financial Provisioning) Act* 2018 (MERFP Act) that was passed in November 2018. New regulatory requirements result from the reforms and are included here.

Key EA regulatory management issues, particularly in the mine development period, are:

- EA Major Amendment application. The current EA is based on a previous 2016 mine layout. Changes to the mine layout will require submission of an EA Major Amendment to the DES. This is a straightforward requirement with application preparation and pre-lodgement meetings.
- Progressive Rehabilitation and Closure (PRC) plan submission. Organizations carrying out mining activities in Queensland are legally obligated to rehabilitate the land. Recent legislation

reforms require holders of an existing EA for a mining activity relating to a mining lease approved through a site-specific application granted prior to passage of the PRC plan legislation (as per Eva), to develop and submit a PRC plan to the DES. As mine development at Eva has not commenced, a PRC plan is required to be submitted in conjunction with the proposed EA Major Amendment application.

- Estimated Rehabilitation Cost (ERC) decision. An ERC decision is required to be in effect before commencing any activities under an EA. The ERC is the estimated cost of rehabilitating the land on which a resource activity is carried out, and preventing or minimizing environmental harm, or rehabilitating or restoring the environment in relation to the resource activity. DES is responsible for deciding the ERC for an EA for resource activities. The ERC came into effect in 2019 under the MERFP Act reforms, and replaces the previous Plan of Operations (PoO) requirements.
- ERC scheme Financial Assurance (FA). This is required to be lodged with DES (either as a contribution paid to the scheme fund, or as a surety given under the MERFP Act) prior to any activities being allowed to commence. The amount of the FA required is calculated in accordance with DES procedures, based on the implementation of site-specific rehabilitation and closure tasks, using independent contractor third-party rates. The amount of the FA is directly related to the activities authorized.
- Design plan for the Cabbage Tree Creek diversion. Final detailed plans will need to be formally submitted, and approval received, prior to construction being allowed to commence.
- Environmental offset requirements. The Project triggers the requirement of an offset due to the disturbance of regional ecosystems resulting from the disturbance of Cabbage Tree Creek. There are two options for offsets: a financial settlement, or a proponent-driven offset which may include approved conservation work programs. A series of submissions are required, including Significant Impact Details, Offset Report, and Notice of Election at least four months prior to commencement of any site work (Significant Residual Impacts). To fulfil its obligations, the Company intends to opt for a financial settlement, but is interested in investigating a proponent driven offset (at least in part) involving the rehabilitation of Cabbage Tree Creek utilizing an indigenous contractor.

1.5 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

The Project tenements are in North West Queensland, 76 km northwest of the town of Cloncurry and 194 km (by road) northeast of Mount Isa, a regional mining centre. Current site access is by way of gravel roads from a sealed road that passes 8.5 km to the east of the proposed plant site. The site is also 11 km north of the major operating Dugald River zinc mine, owned by MMG.

The town of Cloncurry is located on the railway line from Townsville to Mount Isa, and has container handling facilities, an airport that hosts both commercial and fly-in/fly-out (FIFO) jet aircraft services, and a regional fuel depot. It also has schools, hospitals, and other services. The Project lies within the Shire of Cloncurry, which is the local government administrative area. The Shire offices are also based in Cloncurry.

Grid power is generated in Mount Isa at two gas-fired power stations, and is transmitted from Mount Isa to Cloncurry. A 220 kV power line has been constructed from the Chumvale substation near Cloncurry to the Dugald River mine. CMMC received a term sheet from CopperString, the proponent for developing a high voltage electricity transmission line to connect electricity users in the North

West Minerals Province (NWMP) and the Mount Isa region to the National Electricity Market (NEM) at Woodstock near Townsville. This study allowed for power supply from Mount Isa for Years 1 to 3, and from CopperString from Year 4 onwards.

The Cloncurry region is semi-arid, with a distinct hot, wet season from November to March, which is typical of inland northern Australia. Average monthly temperatures range from 10.6°C to 38.5°C, with extremes recorded from 1.8°C to 46.9°C. Rainfall in the wet season largely occurs as storms. Rainfall is highly variable from year to year, with the region often experiencing both multi-year droughts and large-scale flooding from major rainfall events.

The Project site is serviced by a complex system of surface drainages that flow generally northward. On the western side of the processing plant and Little Eva pit is Cabbage Tree Creek, which is joined by other creeks flowing northward to become a tributary of the Leichhardt River. Creeks and rivers flow only during, and for a brief period following, the wet season.

The Project has groundwater sources from both hard rock fracture zone systems and from a graben-like structure infilled with Phanerozoic sediments and alluvial deposits within a paleodrainage adjacent to the current course of Cabbage Tree Creek.

The mine site and broader operation area is gently undulating flat topography, following a discrete north-south ridgeline that transects the area on the western side of the Bedford pit. The site is currently crossed by several gravel roads from pastoral and exploration activities. SunWater Limited (SunWater)'s water pipeline from Lake Julius to the Ernest Henry mine crosses the lease area from west to east. The predominant land use is low-intensity cattle grazing, although exploration and mining activities have been conducted over the area since the late 1800s.

1.6 History

The Project has a long history, and has been held under various tenures by a variety of exploration and mining companies. Small-scale mining dating back to the early 1900s has occurred at deposits such as Little Eva, Bedford, and Lady Clayre. Early explorers that contributed significantly to the Project with the discovery of the copper-only or native copper deposits are Ausminda Pty. Ltd., and then CRA Exploration (CRAE), who completed the first substantive work between 1990 and 1996, also defining a small resource at Little Eva. CRAE sold its interest in the Project to Pasminco in 1998. Altona acquired the Project in 2001. Altona purchased the tenement hosting the Ivy Ann deposit from Dominion Metals Pty. Ltd. (Dominion) and Pan Australian Resources NL (PanAust).

The remaining property was acquired by purchasing tenure from both Pasminco and Lake Gold Pty. Ltd. in a 50:50 ownership split between Altona and Roseby Copper Pty. Ltd. (RCPL). In 2004, Altona purchased RCPL, and thus Altona held 100% of the Eva Copper Project resources. Until 2009, work focused extensively on the copper-only resources, with completion of two feasibility studies based on blends of sulphide ore and copper-only ore. From 2009 to 2012, Altona carried out additional drilling, resulting in Mineral Resource upgrades at the Little Eva, Bedford, Lady Clayre, Ivy Ann, Blackard, Legend, and Scanlan deposits. Little Eva's resource estimate was doubled due to the additional drilling.

In 2012, Altona completed a Feasibility Study based on the increased resources at the copper-gold sulphide deposits, and excluding the Blackard and Scanlan deposits. Altona published Mineral Reserves for the Little Eva, Bedford, Lady Clayre, and Ivy Ann deposits as part of the 2012 Feasibility

Study. Altona published updates to the Feasibility Study in 2014 and 2017. The 2017 update incorporated the subsequently delineated significant Mineral Resource at Turkey Creek.

MLs and an EA were granted in 2012 based on the 2009 Feasibility Study mine plan. An EA amendment was granted in 2016 based on the revised 2012 Feasibility Study mine plan and the integration of Turkey Creek into that mine plan; this is the current EA.

Altona completed a DFS update in 2017, incorporating the Turkey Creek deposit in the mine plan and significant layout changes that included changes to the size and location of the TSF and a Cabbage Tree Creek diversion channel at Little Eva pit. To support the previous studies, the Little Eva, Bedford, Lady Clayre, and Ivy Ann deposits have had a number of formal Mineral Resource estimates that reflect stages of resource definition dating from 2006 to 2017. The only Mineral Resource estimate for Turkey Creek was completed in 2015. Estimates were largely undertaken by external independent experts, initially by McDonald Speijers, and most recently Optiro, based on data and geological models provided by the Company.

CMMC completed a Feasibility Study in September 2018 in which in-house experts for this study produced revised Mineral Resource and Mineral Reserve estimates. No significant resource drilling was completed since the previous published resource update. However, metallurgical data was collected from existing samples and two new drill holes in the Little Eva pit.

CMMC commenced this report, a Feasibility Study, in 2019, in which in-house experts for this study produced revised Mineral Resource estimates based on additional data and infill drilling at Blackard. Additional metallurgical data was collected from existing samples and from new drill holes at the Little Eva and Turkey Creek pits.

1.7 Geological Setting and Mineralization

The Project area is situated within the Mount Isa and North West Region of Queensland, Australia, an area that is one of the premier base metal-bearing areas of Australia, with mining activities having taken place since the discovery of copper and gold near Cloncurry in the 1860s. The Mount Isa area hosts numerous base metal copper, zinc, and lead deposits of global significance, including the Mount Isa, Ernest Henry, Century, Dugald River, Canington, and Selwyn deposits. The Eva Copper Project is hosted by Proterozoic-aged, metamorphosed and poly-deformed marine sedimentary and volcanic rocks of the Mary Kathleen domain of the Eastern Fold Belt Inlier. Deformation, metamorphism, and plutonic activity took place during the Isan Orogeny, approximately 1,600 to 1,500 million years (Ma) ago.

There are twelve known mineral deposits in the Project area, of which seven have been included in the current mine plan. Mineral deposits are grouped into two types: copper-gold, and copper only. There are five of the copper-gold deposits, all of which are in the mine plan. These deposits are classified as iron oxide copper-gold (IOCG) deposits, where mineralization is associated with regional-scale hematite and albite alteration (red-rock alteration), and localized magnetite alteration. Copper sulphide mineralization, primarily chalcopyrite with lesser bornite, occurs as veins, breccias, fracture fill, and disseminations in mafic to intermediate volcanic or intrusive rocks. Gold is generally correlated with copper, and is recovered in the copper concentrate. Mineralization appears to be localized and/or bounded by faults and other deformation-related structures.

The copper-only deposits are stratabound, locally stratiform, and most occur within metamorphosed calcareous metasedimentary rocks, forming an approximately linear trend stretching over 7 km. The origin of these deposits is uncertain; they may be deformed and metamorphosed versions of sedimentary or red-bed type copper deposits, or they could be more closely related to the IOCG deposits, but with enhanced stratigraphic controls related to the calcareous beds being particularly reactive with hydrothermal fluids.

All of the deposits have a 10 m to 25 m thick overlying zone of oxidation, where the rock is extensively weathered, and copper sulphide minerals have been leached or converted to various oxide minerals that cannot be recovered by flotation. The oxide zones are treated as waste, but tonnages and copper grades have been estimated. With the exception of the Turkey Creek deposit, the copper-only deposits commonly have a significant thickness of supergene material, where carbonate has been leached from the rock, reducing hardness and density, and the copper occurs as native-copper, chalcocite, and other low-sulphur copper species. The carbonate-leached zone is separated from the underlying sulphide zone by a thin transition zone. Each of these mineralogical zones has been modelled so that resources can be estimated for each and the appropriate metallurgical recoveries can be applied for reserve estimation.

1.8 Drilling

Although exploration work has been recorded within the Eva Copper Project area since 1963, usable drill data dates back to 1988. Total drilling in the seven deposits with planned production includes 1,470 drill holes for 208,637 m. All the drill holes used for Mineral Resource estimation have accurate collar and downhole surveys, including the older holes, which were subsequently resurveyed by later exploration companies (Universal, or more recently, Altona). Most of the drilling was done by reverse circulation (RC) methods, with a small percentage being diamond drill holes (DDH). Approximately 50% of the drilling and 30% of the meterage in the Ivy Ann deposit is from percussion holes. Statistical analysis of the type of drilling, age, and operating company does not indicate any bias to the drill hole assay data. Assay data from two DDHs completed by Sichuan Railway Investment Group (SRIG) in 2017, and two DDH completed in 2018 by CMMC within the Little Eva deposit, provided material for metallurgical testing and were used to verify the resource block model. Two holes were drilled in the Turkey Creek deposit in 2018 and 2019 for grade verification and metallurgical material. Eighteen reverse circulation (RC) holes were drilled in the Blackard deposit in 2019 by CMMC to upgrade resource classification. Assay data from the 2019 RC drilling within the Blackard deposit is statistically indistinguishable from historical drilling.

1.9 Exploration

Mineral exploration on lands of the Eva Copper Project dates back more than 40 years. The exploration database for the area contains information from numerous geological, geophysical, and geochemical surveys carried out by the current and previous operators, in addition to regional government data on geology and geophysics. Almost all data from historical geophysical and geochemical work is compiled in the Company database, and has been used in the design and guidance of current exploration work.

The most useful historical geophysical work includes ground and airborne magnetics and gravity surveys which, when combined with soil geochemistry, provide good drill targeting tools. Induced polarization (IP) and electromagnetic (EM) geophysical surveys have also proven to be useful or have some benefit in the right circumstances. Continuous improvements in electronic instrumentation, computer data processing, inversion technology for geophysics, and multi-element analysis (particularly in handheld, portable X-ray fluorescence (XRF) units), provide significant rationale to continue geophysical and geochemical surveying on the property.

1.10 Deposit Types

Copper deposits of the Eva Copper Project are of two types. The most significant are those of the IOCG type, which are hydrothermal copper-gold deposits associated with relatively high contents of iron oxide minerals (magnetite or hematite), a general lack of quartz, and extensive sodic alteration. The hydrothermal fluids are believed to be sourced from, and/or driven by, magmatic systems with possible addition of basin brines; however, mineralization is commonly distal (or spatially distinct) from the causative plutonic rocks. Mineralization can take many forms, but the dominant ones are vein networks, breccias, dissemination, and replacement. Both structure (fault or fracture systems) and lithology (chemistry and rheology) are key features in localization of mineralization. The second type of copper deposit is termed copper-only; these deposits do not contain significant gold, and are typically hosted within deformed and metamorphosed calcareous sedimentary rocks as stratabound mineralization. One deposit, Turkey Creek, is a stratabound copper-only deposit within volcanic rocks, and has processing characteristics similar to those for the copper-gold deposits.

There are 12 defined deposits within the Eva Copper Project, ranging in size from 0.7 Mt to over 100 Mt, seven of which are included within the current mine plan. Four are copper-gold deposits, and three are copper-only deposits. Metallurgical recoveries for the copper-gold deposits are favourable, due to relatively coarse-grained chalcopyrite and lesser bornite. All of the deposits have a thin, 10 m to 40 m weathered or oxide zone at surface, for which tonnage and grades have been estimated, but which have been treated as waste within the mine plan. The copper-only deposits hosted within calcareous metasedimentary rocks have additional zones of weathering and/or acid leaching, which has removed carbonate, reducing rock strength and density in addition to changing sulphide mineralogy. In the two such deposits, Blackard and Scanlan, a supergene zone termed native copper occurs below the oxide zone, and contains abundant native copper in addition to chalcocite, cuprite, and other low-sulphur copper species. Extensive metallurgical testing has been carried out on these deposits, with appropriate processing design and estimation of recoveries. Within these deposits a narrow transition zone occurs between the copper zone and underlying sulphide zone.

1.11 Sample Preparation, Analyses, and Security

There is very little documentation about sample collection, preparation, and security for the pre-1997 drilling campaigns, although the nature of the exploration programs, preservation of data, and logging records all indicate that the drilling programs were carried out in a professional and competent manner. Later exploration programs by Universal (beginning in 2002) and Altona (in 2011), which provided the vast majority of the drill data, were carried out with above industry-standard sample collection methods, and appropriate quality assurance and quality control (QA/QC) protocols. RC drilling accounts for more than 90% of the Project samples, and these samples were collected using standard cyclones and splitters at the drill site. Samples lengths were initially 2 m for Universal; however, they were changed to 1 m in 2003. Almost all of Altona’s samples were 1 m in length. Samples were bagged and sealed in the field, and shipped to commercial laboratories in either Townsville or Brisbane. Regular duplicate samples of RC chips were inserted into the sample stream at a rate of 1 in every 20, and triplicate samples collected at the time of drilling were inserted into the sample stream at the rate of 1 in every 40. Appropriate reference standards and blank samples were inserted at rates of 1 in every 20 and 1 in every 45, respectively. Much of the sample material has been retained, mostly as pulp samples; however, there is some coarse reject material, and it is stored in carefully organized warehouses, which also contain split diamond drill core. All analytical information has been carefully archived in an electronic database, which has been reviewed for accuracy by independent consultants and CMMC.

1.12 Data Verification

Historical drill locations were checked and resurveyed by subsequent operators, and assay data has been examined and checked by third-party consultants involved in previous Feasibility Studies. There is no apparent bias in the assay data from drill campaigns involving four different companies. The resource QP examined drill core on site and found good agreement between geology and historical logs, and visual estimates of copper grade were in agreement with assays. Assay results from drill holes completed to obtain metallurgical samples in the Little Eva and Turkey Creek deposits in 2018, and in the Blackard deposit in 2019, compare favourably to adjacent block grades within the block model, supporting both the database and Mineral Resource estimation.

1.13 Metallurgical Testwork and Process Design

This section summarizes both historical and recent testwork associated with the various ore types on the Project property. For additional information, reference the 2018 Feasibility Study completed by Hatch for CMMC in 2018, the GR Engineering Services (GRES) Definitive Feasibility Study (DFS) for Altona in 2014, and the GRES DFS for Universal in 2009. The previous Feasibility Studies discuss in detail the metallurgical performance of ores from the Little Eva pit and associated satellite pits, which contain classic, flotation-amenable copper sulphide ore types. Work completed as part of the present Feasibility Study expands upon the previous Feasibility Studies and considers the addition of other pits, including those containing native copper-bearing reserves which require more unique processing approaches, as had been the focus of the earlier 2009 DFS. This report generalizes the various ore sources into one of two classes for design purposes: sulphides, and native copper. The various ore sources were studied from the perspective of newer technologies, including HPGR for comminution, and direct flotation reactors (DFR) for flotation.

The Little Eva pit is the main ore source for the Project, containing 97.7 Mt at 0.38% Cu. This pit has been well studied, with 145 flotation tests from multiple core and RC chip sources that ranged in scope from benchtop to pilot plant. This ore consistently demonstrates high recovery performance with a high degree of liberation at relatively coarse grinds. The average ore competency lies near the 50th percentile of the JK database, with medium to hard Bond work indices. Copper is present as chalcopyrite with trace amounts of pyrite. Strong flotation kinetics result in high recoveries, concentrating to a good final concentrate grade following a nominal regrind with no pH modification. Overall, this ore type presents low technical risk.

The sulphide satellite pits, comprising Turkey Creek, Bedford, Lady Clayre, and Ivy Ann, are smaller sources, together representing 19.4 Mt of the overall reserve. These ore types are generally similar to Little Eva from both a comminution and flotation perspective. Some differences include a stronger deportment of copper to bornite, and varying grade distribution. Overall, these pits show average copper recoveries of 88% to 95%, and represent high-grade sources of high recovery material. The specific recoveries for each pit are used as inputs into the mine schedule and financial model.

The copper-only pits, Blackard and Scanlan, are distinctly different from other pits in the area, containing oxide cap, native copper, sulphide transition, and sulphide zones. Combined, these pits represent 53.8 Mt of ore. The native copper zones are the largest copper-bearing zones within these pits, containing a relatively fine distribution of native copper with varying quantities of sulphides. These pits were studied by previous owners; however, several recent updates have been completed. In total, 410 flotation tests (including blended ore feed) have been completed, ranging from benchtop to pilot scale work. On a flotation basis, the native copper zones typically achieve 60% recovery, with an additional 2% to 3% achievable by gravity methods. Recovery is highly variable as deportment shifts from native copper to sulphides, requiring flexibility within the processing flowsheet between gravity and flotation operations to achieve an average of 63% overall recovery. This ore is typically very soft, resulting in low comminution costs and high mill throughputs. Below the native copper-bearing zones of both Blackard and Scanlan are sulphide zones containing bornite and chalcopyrite, behaving similarly to Turkey Creek ore. The flotation response of the ore from the native copper to the sulphide transition zone increases with sulphide content, as expected.

For determining key comminution values for plant design, the 70th percentile of the dataset was used to ensure confidence in comminution equipment sizing. For this feasibility study, Ausenco's proprietary Ausgrind power-based calculation suite was used, which is mainly driven by Dr. Steve Morrell's (SMC testing) parameters and Bond work indices (Lane et al., 2013).

In total, the abovementioned work has been sourced from 25 metallurgical testing campaigns completed at established metallurgical labs throughout Australia and British Columbia, Canada, from 1996 to 2019.

1.13.1 Highlights of Selected Test Results

The following summarizes the main aspects of the test results obtained, which were used as the basis for the process design criteria for the processing plant.

A target ore blend of 75:25 of sulphide ores to native copper ores (Blackard and Scanlan pits only) were selected based on estimated HPGR capacity. The sulphide ore types exhibit average to high ore competency and hardness, whereas the native copper components are considered very soft.

At a target grind of 165 µm, the design plant recovery is 87% using this component ore blend of high recovery sulphide and low recovery native copper. Individual recoveries for each pit were determined based on metallurgical test results, and these were used for the economic model.

The final concentrate grade of 28% is based on locked-cycle and cleaner circuit test results, and represents a reasonable estimate for the final concentrate grade in processing the ore blend. The gravity circuit final product is expected to support this final grade target.

1.13.2 Comminution

Previous Feasibility Studies presented detailed test information for Axb values, Bond work indices (BWi), and uniaxial compressive strength (UCS). Updated SMC Test® and BWi work was performed in 2019 on several of the pits to ensure that the satellite pits are well understood. Bulk samples from Little Eva and Blackard were sent to the Metso York laboratory to determine HPGR performance. This data was used for sizing the HPGR and determining the plant throughput. Due to the lower specific gravity of Blackard, a 25% ore blend was selected to ensure high HPGR throughput, as this is a volumetric machine.

The full datasets of Little Eva and Blackard were used for determining inputs used in the plant design. The 70th percentile of available data was used by Ausenco in their Ausgrind tool for comminution circuit design.

1.13.3 Gravity and Flotation Recovery

In 2019, samples of Blackard ore were sent to Process Mineralogical Consulting (PMC) in Maple Ridge, British Columbia, to confirm historical mineralogical reports. The results indicated a high-grade gravity concentrate was possible with native copper present at an average grain size of 100 µm. The coarse fractions were heavily deported towards native copper, with increasing deportment of sulphides in the finer fractions. Cuprite was present in complex particles with both native copper and sulphides.

A separate Blackard composite was sent to Gekko Systems Pty Ltd (Gekko) located in Ballarat, Victoria, Australia, in 2019, to assess generating a saleable final concentrate generated by gravity operations alone. This testwork showed very high-grade concentrates can be generated when feeding only native copper sources, and greater than 28% concentrates can be generated when processing feed streams blended with sulphide ore types. Also, in 2019, bulk samples from Little Eva, Turkey Creek, and Blackard were sent to the Copper Mountain Mine metallurgical laboratory, where a pilot scale direct flotation reactor (DFR) was set up to test the viability of the technology. The results were in line with historical recoveries, with the DFRs showing higher selectivity on the rougher stage. However, DFR cells were not selected for the rougher circuit due to concern over risk associated with new technologies; DFRs were selected for the cleaner circuit only, based on positive results seen with the pilot cell operating on Copper Mountain in-stream rougher concentrate.

Updated testwork was performed on Turkey Creek composites at Australian Laboratory Services (ALS) in Perth. The deposit is described geologically as having two separate zones, named the “upper” and “lower” zones. The testwork confirmed that recovery performance is the same for both of these zones, which is in line with historical testwork.

Samples from the Blackard sulphide zone were obtained, with a master composite being sent to Base Metallurgical Laboratories (Base Met) in Kamloops, British Columbia. The testwork confirmed strong recovery performance, more in line with that of Turkey Creek, showing a stronger copper deportment towards bornite.

Additional samples from the sulphide deposits were sent to ALS in Perth, and the results were in line with historical recoveries.

1.13.4 Concentrate Characterization

Detailed chemical analyses were performed on the concentrates produced from the testwork programs, and the results indicate that there appear to be no impurity elements present in the concentrate at a level that will incur smelter penalties. Provision for separate dewatering and containment of gravity concentrates is included in the plant design for future sampling or marketing opportunities.

1.13.5 Tailings Handling

Tailings generated from the bulk samples processed during the DFR testwork were sent to Paterson & Cooke in Denver, Colorado, for tailings characterization. The samples were examined both separately and as a blend. In both cases no concerns were highlighted with tailings settling performance. A reasonable target of 63% solids was selected for tailings thickener underflow design.

1.14 Mineral Resources Estimate

1.14.1 Eva Copper Project Resources

Mineral Resource estimates were prepared by CMMC personnel, based on all drilling conducted up to October 2019. The effective date of the resource estimates is January 31, 2020. Only the Little Eva, Turkey Creek, and Blackard deposits have new data, which is limited to two core holes in the Little Eva deposit, one in Turkey Creek, and eighteen RC holes in the Blackard deposit. The new drilling was primarily to obtain material for metallurgical testing, but additionally for the verification of grades in the historical data, and infill drilling at the Blackard deposit. Resource estimates by CMMC have been completed on all deposits within the mine plan. The resource estimates were made using methods and block sizes deemed appropriate for the anticipated mining methods, mining equipment, and grade control methods described in this report. The constraining pit shells for defining the limits of Inferred resources are based on economic values that are, among other inputs, dependent upon metallurgical recoveries which have been determined from work carried out, and described, in this report. Resources were constrained by Whittle pit shells for the Little Eva and Turkey Creek deposits generated using metal prices of $3.50/lb Cu and $1,250/oz Au. Pit shell constraints for the other deposits were generated using metal prices of $3.50/lb for copper and $1,250/oz for gold.

A zone of oxidation overlies all of the seven deposits in the Eva Copper Project. The base of the oxidized zone is generally sharp (±2 m), and was modelled during resource estimation. In the current mine plan, the oxidized material is treated as waste, as currently there does not appear to be any form of economic extraction; however, grades have been modelled and tonnages tabulated for general interest and in the event of possible processing in the future. The tonnage and grade of oxidized material were determined in the same manner and at the same time as the other resource estimations.

The two copper-only deposits in the mine plan, Blackard and Scanlan, were not included in CMMC's previous study, as processing methods and recoveries were uncertain. Metallurgical testing was completed in 2019 on mineralization from the Blackard deposit, which has resulted in determination of a suitable process methodology with reliable recovery estimates, such that these deposits are now included in the mine plan.

Table 1-2: Eva Copper Project Mineral Resources, January 31, 2020

	Tonnes (kt)	Cu Grade (% Cu)	Au Grade (g/t)	Cu Pounds (Mlb)	Au Ounces (koz)
Measured					
Little Eva	56,671	0.39	0.07	492	129
Turkey Creek	6,938	0.47	-	72	-
Blackard*	30,595	0.51	-	343	-
Scanlan*	11,397	0.59	-	147	-
Bedford	-	-			
Lady Clayre	5,113	0.42	0.17	47	28
Ivy Ann	1,107	0.38	0.07	9	3
Total Measured	**111,821**	**0.45**	**0.05**	**1,110**	**160**
Indicated					
Little Eva	65,154	0.34	0.07	486	135
Turkey Creek	6,871	0.44	-	67	-
Blackard*	53,073	0.45	-	521	=
Scanlan*	14,453	0.46	-	146	-
Bedford	3,002	0.54	0.14	36	14
Lady Clayre	2,228	0.40	0.18	20	13
Ivy Ann	4,037	0.35	0.08	31	10
Total Indicated	**148,818**	**0.40**	**0.04**	**1,307**	**172**
Measured + Indicated					
Little Eva	121,826	0.36	0.07	978	264
Turkey Creek	13,808	0.46	-	140	-
Blackard*	83,688	0.47	-	864	
Scanlan*	25,850	0.52	-	294	-
Bedford	3,002	0.54	0.14	36	14
Lady Clayre	7,341	0.41	0.17	66	40
Ivy Ann	5,144	0.36	0.08	41	13
Total Measured + Indicated	**260,659**	**0.42**	**0.04**	**2,419**	**330**
Inferred					
Little Eva	3,764	0.31	0.07	26	23
Turkey Creek	12,897	0.40	-	113	-
Blackard*	19,457	0.48	-	207	-
Scanlan*	3,432	0.44	-	33	-
Bedford	792	0.42	0.14	7	3
Lady Clayre	4,964	0.36	0.15	40	23
Ivy Ann	961	0.32	0.07	7	2
Total Inferred	**46,267**	**0.42**	**0.04**	**431**	**51**

Notes: *****Blackard and Scanlan deposit cut-off grades are based on NSR values which vary by mineralogical zone to reflect estimated recoveries and distance from the processing plant. Copper cut-off grades for the low-, mid-, and high-grade cut-offs are provided in Table 14-32.

Mineral Resources:

1. Joint Ore Reserves Code (JORC) and CIM definitions were followed for Mineral Resources.

2. Mineral Resources are inclusive of Mineral Reserves.

3. Mineral Resources are constrained within a Whittle pit shell generated with a copper price of $3.50/lb, a gold price of $1,250/oz and an exchange rate of AU$1.35 = US$1.00.

4. Density measurements were applied (ranges from 2.4 t/m^3 to 3.0 t/m^3).

5. Significant figures have been reduced to reflect uncertainty of estimations and therefore numbers may not add due to rounding.

1.14.2 Other Deposits Historical Resources

In addition to the Blackard and Scanlan deposits, there are five additional copper-only deposits that occur along an approximately linear trend, extending from the Legend deposit in the north to the Lady Clayre deposit in the south. In general, other than Blackard and Scanlan, the historical resource estimates for these deposits are relatively small. However, most of the deposits remain open to expansion, and in particular the Legend deposit, which is the northern extension of the Blackard deposit, is proximal to mine infrastructure. The Company has not estimated resources in these deposits, and the values presented in Table 1-3 are historical estimates only.

Table 1-3: Eva Copper Project – Historical Mineral Resources, Copper-Only Deposits

Deposit	Tonnes (kt)	Cu Grade (%)	Au Grade (g/t)	Cu Pounds (Mlb)	Au Ounces (koz)
Legend	17,400	0.54	0	207	0
Great Southern	6,000	0.61	0	81	0
Longamundi	10,400	0.66	0	151	0
Caroline	3,600	0.53	0	42	0
Charlie Brown	700	0.40	0	6	0
Total	**38,100**	**0.58**	**0**	**487**	**0**

Notes: **1.** Historical Mineral Resources reported by Altona, in accordance with JORC (2012), for their 2017 DFS. **2.** The historical Mineral Resources cannot be relied upon until further due diligence is completed by CMMC. **3.** Historical resources reported above a 0.30% Cu cut-off grade. **4.** Totals may not add due to rounding.

1.15 Mineral Reserve Estimate

The Eva Copper Project has a Mineral Reserve of 171 Mt grading 0.46% Cu and 0.05 g/t Au for 1.718 billion pounds (Blb) contained copper, and 260,000 oz contained gold. Approximately 95% of the Mineral Reserve is contained in the Little Eva, Blackard, Scanlan, and Turkey Creek deposits. The Bedford, Lady Clayre, and Ivy Ann satellite deposits compose the remaining 5% of the Mineral Reserves. Little Eva and Turkey Creek will be mined first, and the satellite deposits will supplement the Project's production in the latter years. Approximately 25% of the mill feed will now be softer native copper ores, originating from the Blackard and Scanlan deposit areas.

All deposits have ore tonnages classified as either Proven or Probable Mineral Reserves only, and additional Inferred Mineral Resources are not included in the mine schedule. The Mineral Reserve is summarized in Table 1-4.

All Mineral Reserves are classified and reported in accordance with the 2011 CIM Standard. CMMC considers the Mineral Reserve estimate, checked by QP Stuart Collins, P.E., to be reasonable, acceptable, and reported in accordance with CIM definitions and NI 43-101.

The Mineral Reserves are generated based on the mine designs applied to the Measured and Indicated Mineral Resources only. The design methodology uses both the cut-off grade estimation and economic assessment to design and validate the Mineral Reserves. CMMC is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Mineral Reserve estimate.

Table 1-4: Eva Copper Project Mineral Reserves, January 31, 2020

Deposit	Mineral Reserve Classification	Cut-off Value (US$/t)	Ore Tonnes (kt)	Cu Grade (% Cu)	Au Grade (g/t)	Total Cu Pounds (Mlb)	Total Au Ounces (koz)
Little Eva	Proven	8.95	53,907	0.40	0.07	480	126
Lady Clayre	Proven	10.32	2,648	0.46	0.19	27	16
Ivy Ann	Proven	11.44	685	0.44	0.09	7	2
Bedford	Proven	9.35				-	-
Blackard	Proven	9.35	22,951	0.58		295	-
Scanlan	Proven	10.32	6,279	0.72		100	-
Turkey Creek	Proven	8.95	6,151	0.49		66	-
Total	**Proven**	**Varies**	**92,623**	**0.48**	**0.05**	**975**	**144**
Total Gold Grade only	**Proven**		57,241	0.41	0.08	513	144
Little Eva	Probable	8.95	43,805	0.36	0.06	348	91
Lady Clayre	Probable	10.32	831	0.45	0.21	8	6
Ivy Ann	Probable	11.44	1,640	0.42	0.09	15	5
Bedford	Probable	9.35	2,863	0.56	0.15	35	14
Blackard	Probable	9.35	19,756	0.52		228	-
Scanlan	Probable	10.32	4,987	0.58		64	-
Turkey Creek	Probable	8.95	4,544	0.45		45	-
Total	**Probable**	**Varies**	**78,425**	**0.43**	**0.05**	**743**	**115**
Total Gold Grade only	**Probable**		49,139	0.37	0.07	406	115
Little Eva	Proven + Probable	8.95	97,712	0.38	0.07	828	217
Lady Clayre	Proven + Probable	10.32	3,479	0.45	0.20	35	22
Ivy Ann	Proven + Probable	11.44	2,325	0.43	0.09	22	7
Bedford	Proven + Probable	9.35	2,863	0.56	0.15	35	14
Blackard	Proven + Probable	9.35	42,707	0.56	0.00	523	-
Scanlan	Proven + Probable	10.32	11,266	0.66	0.00	164	-
Turkey Creek	Proven + Probable	8.95	10,695	0.47	0.00	112	-
Total	**Proven + Probable**	**Varies**	**171,047**	**0.46**	**0.05**	**1,718**	**260**
Total Gold Grade Only	**Proven + Probable**		106,380	0.39	0.08	919	260

Notes: **1.** CIM Definition Standards were followed for Mineral Reserves. **2.** Mineral Reserves were generated using the January 31, 2019 mining surface. **3.** Mineral Reserves are reported at an NSR cut-off value of $8.95/t for Little Eva and Turkey Creek, $9.35/t for Bedford and Blackard, $10.32/t for Lady Clayre and Scanlan, and $11.44/t for Ivy Ann. **4.** Mineral Reserves are reported using long-term copper and gold prices of $2.75/lb and $1,250/oz, respectively. **5.** Average process recoveries used in pit optimization ranged from 90% to 93% for copper sulphide, 63% for native copper, and 78% for gold were used for all deposit areas. **6.** Little Eva, Turkey Creek, Bedford, and Lady Clayre have an equivalent 5.3% NSR royalty; Ivy Ann has an equivalent 5.8% royalty. **7.** Blackard, Scanlan, and Turkey Creek do not contain gold. **8.** Totals may show apparent differences due to rounding.

Rounding may result in apparent differences when summing tonnes, grades, and contained metal content. Tonnage and grade measurements are in metric units. Gold grades are reported in grams per tonne (g/t), and copper grades are reported in percent of total copper (%Cu). All oxide material was considered as waste; however, CMMC will take the necessary actions to segregate this material for future processing.

George Orr and Associates conducted a full stability analysis of an earlier planned Little Eva pit based on geotechnical analysis of 21 oriented DDHs covering both an earlier starter-pit design and the final pit design utilized in this study. The northwest portion of the deposit has poor to moderate ground conditions; however, the majority of the planned pit ground conditions are good to moderate. Overall slope angles of 43 degrees, inclusive of pit ramps, have been recommended and are used in the Little Eva pit design. The eastern pit wall has the best ground conditions, and therefore all access ramps have been placed on this wall.

Pit optimization was completed by CMMC and verified by Stuart Collins, P.E. The metallurgical recoveries used in optimization were derived by GRES and OZMET Metallurgical Consultants (OZMET) from all pre-existing testwork carried out by ALS Ammtec in 2011 and 2012, and updated by CMMC in 2019. Metallurgical, economic, and other assumptions were current in 2017, and were updated by CMMC in 2018. These optimizations formed the basis of pit designs and the Mineral Reserves.

The Little Eva mine design includes a 22-m wide dual lane in-pit haul road at a 10% gradient on the east wall of the final pit. The pit is approximately 1,700-m long, 950-m wide, and 310-m deep.

Mining dilution was accounted for in the modelling of the larger size blocks (5 m by 5 m by 5 m). This block size reflects the large-scale bulk nature of the deposit. The degree of selectivity in mining is relatively low, and varies in differing domains of the deposit. Little Eva and Turkey Creek Mineral Reserve mine design is based upon a minimum mining unit of 5 m by 5 m by 5 m. Mine equipment has been scaled to allow selective mining for this size. Mineral Reserves will be classified in grade control either as run-of-mine (ROM) feed to be sent directly to the processing plant, marginal ore to be sent to a stockpile for later treatment, or waste. The opportunity exists to improve grade control and reduce unit mining costs.

Optimization of the Blackard, Scanlan, Bedford, Lady Clayre, and Ivy Ann deposits was completed using inputs similar to those used at Little Eva; however, it was assumed that fixed costs were covered by the Little Eva mine, and the cost of haulage to the processing plant was added to each of the satellite deposit ore processing costs. The distances from Blackard and Bedford, Scanlan and Lady Clayre, and Ivy Ann to the processing plant located near Little Eva and Turkey Creek are 6 km, 20 km, and 36 km, respectively. Metallurgical testwork on these deposits indicates that metallurgical characteristics and recoveries are not materially different from the Little Eva deposit. Scheduling of ore extraction from the satellite deposits will mainly commence in Year 3 and continue through the end of the mine life. Marginal material from the satellite pits will be assumed as waste, and will not be transported to the processing plant.

Pits at the other satellite deposits were designed to the same level of detail as Little Eva and Turkey Creek, and the contribution of the other satellites (Blackard, Scanlan, Lady Clayre, Bedford, and Ivy Ann) has grown to 45% of the Mineral Reserves. New pit optimizations and designs will be completed as new Mineral Resource estimates and geotechnical models become available during the Project's development period.

1.16 Mining Method

Conventional open pit mining methods, which include drilling, blasting, loading, and hauling, will be employed at the Eva Copper Project open pits. The Eva Copper Project is estimated to have a two-year construction period, one of which is pre-production mining. Mining activities are based on open pit mining of the Little Eva deposit at a rate of 31,200 t/d of ore. This primary pit at Little Eva will be supplemented by progressively mining six satellite pit areas at Blackard, Scanlan, Turkey Creek, Bedford North and South, Lady Clayre, and Ivy Ann, to achieve a minimum 11.4 Mt/a mill feed rate.

The mining method involves a 13.4 Mt pre-strip of a starter pit at Little Eva, which includes 1.2 Mt of ore. To sustain a 31,200 t/d production rate during the mine life, stripping will continue at slightly elevated rates for several months after production commences. There will be three pushback pits in Little Eva, three pushbacks at Blackard, and two pushback pits in Turkey Creek, while Bedford, Scanlan, Lady Clayre, and Ivy Ann will have one phase of mining.

Drilling will be carried out using conventional drill and blast (D&B) blasthole drills with diesel-powered front shovel excavation, and off-highway dump truck haulage. The initial main mining fleet consists of two front shovels with 22-m^3 buckets and an operating weight of 400-tonnes each, matched to fourteen (Year -2 and Year -1) 141-tonne off-highway rear dump trucks. This fleet is supplemented by the standard support equipment composed of, but not limited to, track dozers, water trucks, graders, front-end loaders (FELs), light vehicles, and service equipment.

Ore haulage from the Scanlan and Lady Clayre satellite pits will be accomplished with the same mining fleet as discussed above.

Approximately 381 Mt of mine waste will be transported to dumps adjacent to each of the pits, or to the TSF for construction. The TSF is expected to require approximately 65 Mt of mine waste. Waste will also be used to construct an engineered creek diversion channel and flood protection bund around the Little Eva pit, known as the Cabbage Tree Creek (CTC) Bund. The channel and bund will redirect wet season water flows in Cabbage Tree Creek away from the Little Eva pit. Diversion bunds and ditches will also be built around the other open pits, where needed.

The ROM ore will be delivered to the ROM pad, where there will be the capability to direct feed from mine trucks to a gyratory crusher with 600 kW of installed power capable of accepting 1-m diameter rock at a rate of 1,733 t/h (75% crusher availability).

The current mining schedule then prioritizes the mining of ore sequentially from Little Eva, Blackard, Scanlan, and Turkey Creek. The other satellite deposits (Lady Clayre, Bedford, and Ivy Ann), which only account for 5% of the Mineral Reserves, will commence mining towards the middle to end of the mine life. The proximity of Turkey Creek to the mill makes it preferable to mine it early in the mining schedule. Further investigation and rescheduling will be carried out prior to project commencement. Mining of ore from the Bedford pits (North and South) is scheduled to commence in Year 4 and Year 5. Lady Clayre pits are scheduled to be mined in years six to eight, and Ivy Ann in Year 5 through Year 6. As noted previously, three of the satellite pits are quite small compared to the Little Eva and Blackard pits.

Dewatering of the open pits will be required. A plan of dewatering wells, horizontal drains, and sumps is envisioned. A detailed plan will be developed during the Project's development period. It has been estimated that the Little Eva pit dewatering will discharge approximately 4,000 m^3/d, and the Blackard pit dewatering approximately 2,000 m^3/d. This water is slated to be used as make-up water in the processing plant.

The mining schedule and schedule of production of copper in concentrate is shown in Figure 1-2.

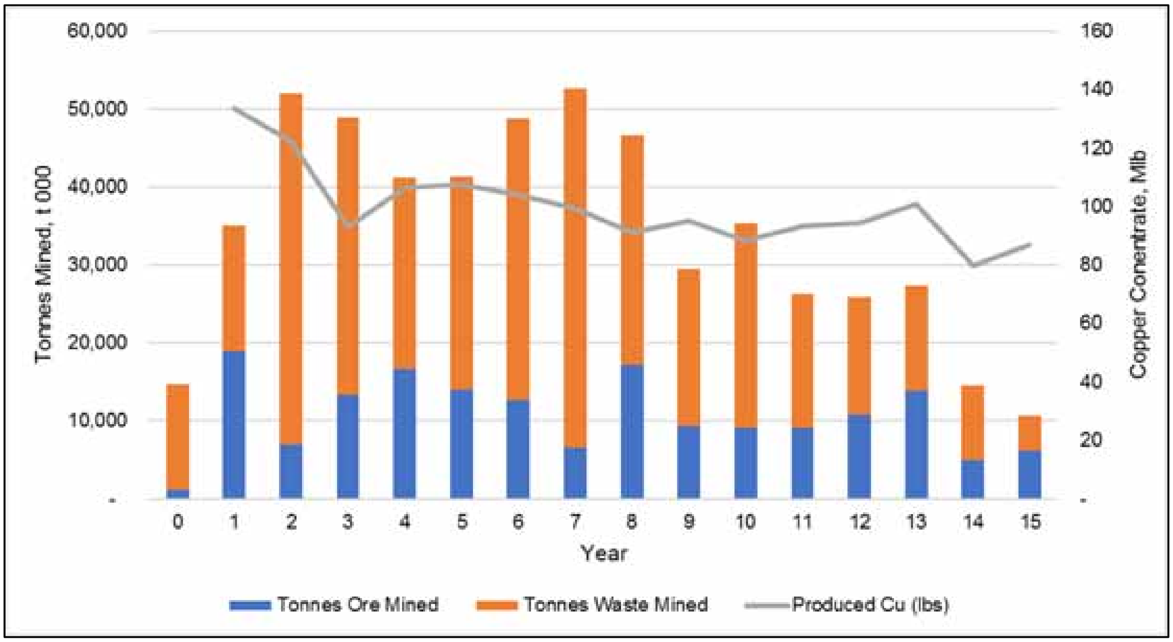


Figure 1-2: LOM Schedule (kt/a) and Copper Production (Mlb/a)

1.16.1 Life-of-Mine and Process Production Schedules

Mining will deliver a nominal 11.388 Mt/a of approximately 0.46% Cu and 0.05 g/t Au ROM feed to the processing plant over a 15-year mine life. Table 1-5 is a summary of the Eva Copper Project's LOM mining schedule. Table 1-6 is a summary of the Eva Copper Project's processing schedule.

Table 1-5: LOM Mining Schedule

Category	Unit	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Native Tonnes	t '000s	35,560	-	-	1	3,620	3,302	3,011	2,986	798	2,989	2,922	2,790	2,921	2,961	2,975	2,460	1,824
Native Cu Grade	% Cu	0.62	-	-	0.31	0.57	0.61	0.63	0.66	0.52	0.53	0.56	0.61	0.66	0.68	0.63	0.67	0.74
Native Cu Tonnes	t	220,863	-	-	3	20,610	20,302	18,970	19,706	4,174	15,885	16,354	16,892	19,156	20,231	18,711	16,364	13,508
Transition Tonnes	t '000s	2,734	-	-	-	12	45	256	542	36	136	78	61	124	279	491	674	-
Transition Cu Grade	% Cu	0.55	-	-	-	0.47	0.65	0.55	0.51	0.47	0.60	0.86	0.41	0.49	0.55	0.56	0.55	-
Transition Cu Tonnes	t	15,022	-	-	-	58	291	1,408	2,752	168	812	676	251	611	1,548	2,729	3,718	-
Sulphide Tonnes	t '000s	132,091	1,168	18,908	6,898	9,643	13,285	10,700	9,155	5,701	14,172	6,301	6,186	6,020	7,597	10,628	2,058	3,669
Sulphide Cu Grade	% Cu	0.41	0.51	0.53	0.43	0.41	0.41	0.42	0.41	0.35	0.33	0.39	0.36	0.39	0.36	0.38	0.40	0.50
Sulphide Cu Tonnes	t	543,767	5,920	100,981	29,902	39,343	54,488	45,320	37,177	20,062	46,989	24,454	22,165	23,272	26,983	40,232	8,199	18,278
Total Ore Tonnes	**t '000s**	**170,386**	**1,168**	**18,908**	**6,899**	**13,275**	**16,632**	**13,966**	**12,683**	**6,535**	**17,296**	**9,301**	**9,038**	**9,066**	**10,838**	**14,095**	**5,192**	**5,494**
Total Ore Cu Grade	**% Cu**	**0.46**	**0.51**	**0.53**	**0.43**	**0.45**	**0.45**	**0.47**	**0.47**	**0.37**	**0.37**	**0.45**	**0.43**	**0.47**	**0.45**	**0.44**	**0.54**	**0.58**
Total Ore Cu Tonnes	**t**	**779,653**	**5,920**	**100,981**	**29,904**	**60,010**	**75,081**	**65,699**	**59,636**	**24,404**	**63,686**	**41,484**	**39,308**	**43,038**	**48,762**	**61,672**	**28,281**	**31,786**
Waste Tonnes	t '000s	380,574	13,520	16,132	45,113	35,669	24,541	27,339	36,100	46,185	29,424	20,233	26,265	17,148	15,077	13,245	9,408	5,174
Total Tonnes	**t '000s**	**550,959**	**14,688**	**35,040**	**52,012**	**48,943**	**41,174**	**41,228**	**46,671**	**52,720**	**46,720**	**29,534**	**35,303**	**26,214**	**25,915**	**27,340**	**14,600**	**10,668**
Sulphide Au Grade	g/t	0.05	0.07	0.08	0.02	0.02	0.06	0.06	0.03	0.08	0.07	0.03	0.04	0.05	0.04	0.04	-	-
Sulphide Au Grams	g	8,083,938	83,892	1,523,098	143,398	293,169	1,071,467	820,904	387,155	511,110	1,275,763	279,997	325,854	414,360	395,217	558,553	-	-
Sulphide Au Ounces	oz '000s	260	3	49	5	9	34	26	12	16	41	9	10	13	13	18	-	-

Notes: 1. Includes oxidized, transition, low-grade mineralization, and Inferred Mineral Resources in the waste tonnage. **2.** Proven and Probable Mineral Reserves are included as ore at NSR cut-off values of $8.95/t for Little Eva and Turkey Creek; $9.35/t for the Blackard and Bedford pits, $10.32/t for the Scanlan and Lady Clayre pits, and $11.44/t for Ivy Ann. **3.** Numbers may not add due to rounding.

Table 1-6: LOM Processing Schedule

	Unit	Total Avg.	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Tonnes Ore Mined	kt	170,386	1,168	18,908	6,899	13,275	16,632	13,966	12,683	6,499	17,196	9,359	9,057	9,048	10,740	13,884	4,903	6,168
Tonnes Waste Mined	kt	380,574	13,520	16,132	45,113	35,669	24,541	27,339	36,100	46,221	29,524	20,175	26,246	17,166	15,175	13,456	9,697	4,501
Total Material Mined	**kt**	**550,959**	**14,688**	**35,040**	**52,012**	**48,943**	**41,174**	**41,305**	**48,783**	**52,720**	**46,720**	**29,534**	**35,303**	**26,214**	**25,915**	**27,340**	**14,600**	**10,668**
Stripping Ratio	(w:o)	2.2	11.6	0.9	6.5	2.7	1.5	2.0	2.8	7.1	1.7	2.2	2.9	1.9	1.4	1.0	2.0	0.7
Tonnes Moved	t/d	96,845	79,826	96,000	142,500	133,725	112,805	113,165	133,651	144,044	128,000	80,915	96,720	71,622	71,000	74,903	40,000	30,649
Milling and Production																		
Dry Tonnes Milled	kt	170,386	-	11,388	11,388	11,419	11,388	11,388	11,388	11,419	11,388	11,388	11,388	11,419	11,388	11,388	11,388	10,860
Re-handle Tonnes	kt	31,833	-	1,726	4,843	1,788	-	-	-	4,920	-	2,029	2,331	2,371	648	-	6,485	4,692
Percent Re-handle	%	18	0	15	43	16	0	0	0	43	0	18	20%	21	6	0	57	43
Native Copper Tonnes	kt	35,560	-	-	1	2,833	2,896	2,800	2,986	2,201	2,989	2,922	2,790	2,921	2,961	2,975	2,460	1,824
Native Copper	%		0	0	0	25	25	25	26	19	26	26	25	26	26	26	22	17
Tonnes Milled	t/d		-	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200
Head Grades																		
Head Grade - Cu	Cu%	0.46	-	0.56	0.53	0.44	0.49	0.50	0.49	0.45	0.42	0.44	0.41	0.43	0.44	0.47	0.37	0.42
Head Grade - Au	Au g/t	0.047	-	0.083	0.052	0.028	0.066	0.059	0.028	0.067	0.079	0.039	0.040	0.047	0.037	0.038	0.027	0.019
Model Cu Recovery	Avg. Cu Rec. %	87.1	-	95	92	85	87	87	84	88	86	86	87	86	86	86	86	87
Head Grade - Density	t/m^3	2.6	-	2.8	2.7	2.6	2.6	2.6	2.5	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6
Contained Copper	Mlb	1,715		140.50	133.17	109.91	122.74	124.33	124.18	112.95	105.87	111.01	101.99	108.19	109.61	118.01	92.47	100.52
Recoveries																		
Recovery - Cu	%	87		95	92	85	87	87	84	88	86	86	87	86	86	86	86	87
Recovery - Au	%	78		78	78	78	78	78	78	78	78	78	78	78	78	78	78	78
Produced Metal																		
Produced Cu	Mlb	1,485		133.48	122.46	92.77	106.48	107.60	104.08	99.08	91.20	95.30	88.25	93.06	94.30	100.93	79.89	76.37
Produced Au	koz	203		23.63	14.72	8.05	18.94	16.77	7.93	19.32	22.68	11.08	11.46	13.58	10.64	10.79	7.77	5.18
Concentrate Produced																		
Concentrate Produced	DMT '000s	2,433		216.2	198.4	150.3	172.5	174.3	168.6	160.5	147.7	154.4	143.0	151.2	152.8	163.5	129.4	150.8

	Unit	Total Avg.	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Concentrate Produced	WMT '000s	2,650		236.3	216.8	164.2	188.5	190.5	184.3	175.4	161.5	168.7	156.2	164.8	167.0	178.7	141.4	164.8
Concentrate Grade	%	28		28	28	28	28	28	28	28	28	28	28	28	28	28	28	28
Moisture %	%	8.5		8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5
Payable Metal																		
Payable - Cu	Mlb	1,448		128.71	118.08	89.45	102.68	103.76	100.37	95.539	87.94	91.90	85.01	89.74	90.93	97.33	77.03	89.73
Payable - Au	koz	186		21.74	13.25	7.22	17.43	15.10	7.13	17.73	20.87	9.97	10.31	12.19	9.57	9.71	6.99	6.81

Notes: **1.** Milled tonnes do not include oxidized, low-grade mineralization, or Inferred Mineral Resources. **2.** Contains stockpile and re-handle material. **3.** Copper recoveries of 95% for sulphide and 63% for native copper materials.
Production: (M) = Months, (Q) = Quarters, (Y) = Years, (DMT) = dry metric tonnes, (WMT) = wet metric tonnes

1.17 Recovery Methods

The Little Eva processing plant has been designed to produce a marketable copper concentrate with a grade of 28% Cu and containing about 3 g/t Au at a nominal throughput rate of 31,200 t/d. A key update in this feasibility study is the change from a SAG mill and pebble crushing circuit to a secondary crusher and HPGR design. The ball mill has also been upsized in order to support 31,200 t/d at a target grind of P_{80} of 165 µm.

The process plant flowsheet developed for processing copper ore from the Eva Copper Project is considered a relatively standard processing plant design for the treatment of copper-bearing sulphide mineral material. All the unit operations selected for the plant design are low risk and of proven technology.

1.17.1 Process Design Description

The following summary of the unit process descriptions is based on the nominal ore throughput rate of 31,200 t/d:

- The ROM ore primary crushing circuit will use a gyratory crusher with a nominal crushing rate of 1,733 t/h (1,925 t/h design) at the availability of 75%. The crusher is a Metso model Mk III 42/65. The crusher will be supplied using haul trucks dumping into a 3-truck capacity dump pocket, equipped with an apron feeder transferring ROM to the primary crusher feed. The primary crusher product size is designed as a P_{80} of 137 mm.
- A MP1250 or equivalent secondary crusher will operate in a closed loop with a double deck vibrating screen to produce a product size P_{80} of 35 mm. The screen undersize will be transferred to the fine ore stockpile.
- The fine ore stockpile has a live capacity of 30,086 tonnes. The crushed ore stockpile reclaim system will be equipped with two reclaim apron feeders each capable of supplying 100% of downstream tonnage.
- A 2.4 m by 1.65 m HPGR supplied with two 5.4 MW drives will operate in a closed loop with two 4.2 m by 8.5 m double deck vibrating screens. The target transfer size to the ball mill circuit is a P_{80} of 4 mm. A nominal and design circulating load has been set at 85% and 130% for the conveyors, respectively.
- The ball mill is a 24-ft diameter by 40-ft EGL with 2 x 7 MW motors. The throughput rate of the grinding circuit will be 1,413 t/h (nominal) and 1,700 t/h (design) at an availability of 92%. The Ball Mill grind product size will have a P_{80} value of 165 µm. Two roughers, a cleaner, and re-cleaner Jigs operating on a bleed of the ball mill cyclone feed will generate a coarse, high grade gravity concentrate to be sent to final concentrate.
- The rougher flotation circuit will consist of six 300 m^3 flotation cells. The circuit has an overall nominal residence time of 35 minutes. Provision is designed for sulphidization in the final two stages of rougher flotations. Rougher concentrate will be reground in the regrind circuit. Rougher tailings will be discharged to the 50 m diameter tailings thickener.
- The rougher concentrate regrinding circuit incorporates a Vertimill® (model VTM1500) and cyclones for classification. The regrind circuit product size will have a P_{80} value of 53 µm. A partial underflow stream will be directed to a flash flotation unit producing a final grade copper concentrate with the tailings returned to the regrind mill. A bowl concentrator will operate on a

bleed of cyclone underflow, concentrating free native copper collecting within the regrind cyclone loop. This product will be sent to final concentrate.

- The regrind cyclone overflow will feed two 18 m^3 DFR first cleaners, producing a >28% concentrate sent to the final concentrate thickener. Six 18 m^3 first cleaner scavengers will send concentrate to three 6 m^3 second cleaners. The second cleaner concentrate will join the first cleaners concentrate as final concentrate. Tailings from the second cleaners will be recycled to the regrind mill. Tails from the cleaner scavengers will be sent to final tailings.
- The concentrate thickener will collect the concentrate products from the cleaner and recleaner DFR cells. The thickened concentrate will be delivered to the concentrate storage tank, and this will feed the concentrate filter press. The filtered copper concentrate will have a moisture value of 8.5%. The dewatered final concentrate will be loaded onto trucks for despatch to smelters.
- A separate dewatering cone and drying paddock is included for dewatering of gravity concentrates.
- Under nominal design conditions, the copper concentrate production can be 194 kt/a inclusive of gravity and flotation concentrates.
- The concentrate thickener overflow solution will be recovered and used in the grinding and flotation circuit.
- The tailings thickener will combine the rougher and cleaner scavenger tailings for discharging to the TSF as final tailings. Process water will be recovered from the tailings thickener and the TSF for re-use in the plant.
- The process water circuit will provide process water to the grinding circuit and other parts of the plant.
- A fresh water circuit will provide water for reagent make-up, gland service, mill lube systems cooling water, filter press cloth wash, and process water make-up.
- The reagent preparation section will prepare the flotation collector reagent (PAX) and the frother reagent for distribution to the slurry streams. A liquid sodium hydrosulphide circuit will supply sulphidizer to the final stages of rougher recovery. The flocculant required for the tailings and concentrate thickeners will also be prepared in this section. A “test reagent” circuit is included for the testing of additional reagents.
- Various process streams will be sampled automatically on an on-line basis and analyzed for copper to provide the necessary information for process control and a metallurgical balance.
- An assay and metallurgical laboratory will be included in the design.
- Site services, power supply, air supply, and water supply will be included in the design.

The simplified process flowsheet is shown in Figure 1-3 and a 3D layout view of the processing plant is shown in Figure 1-4.

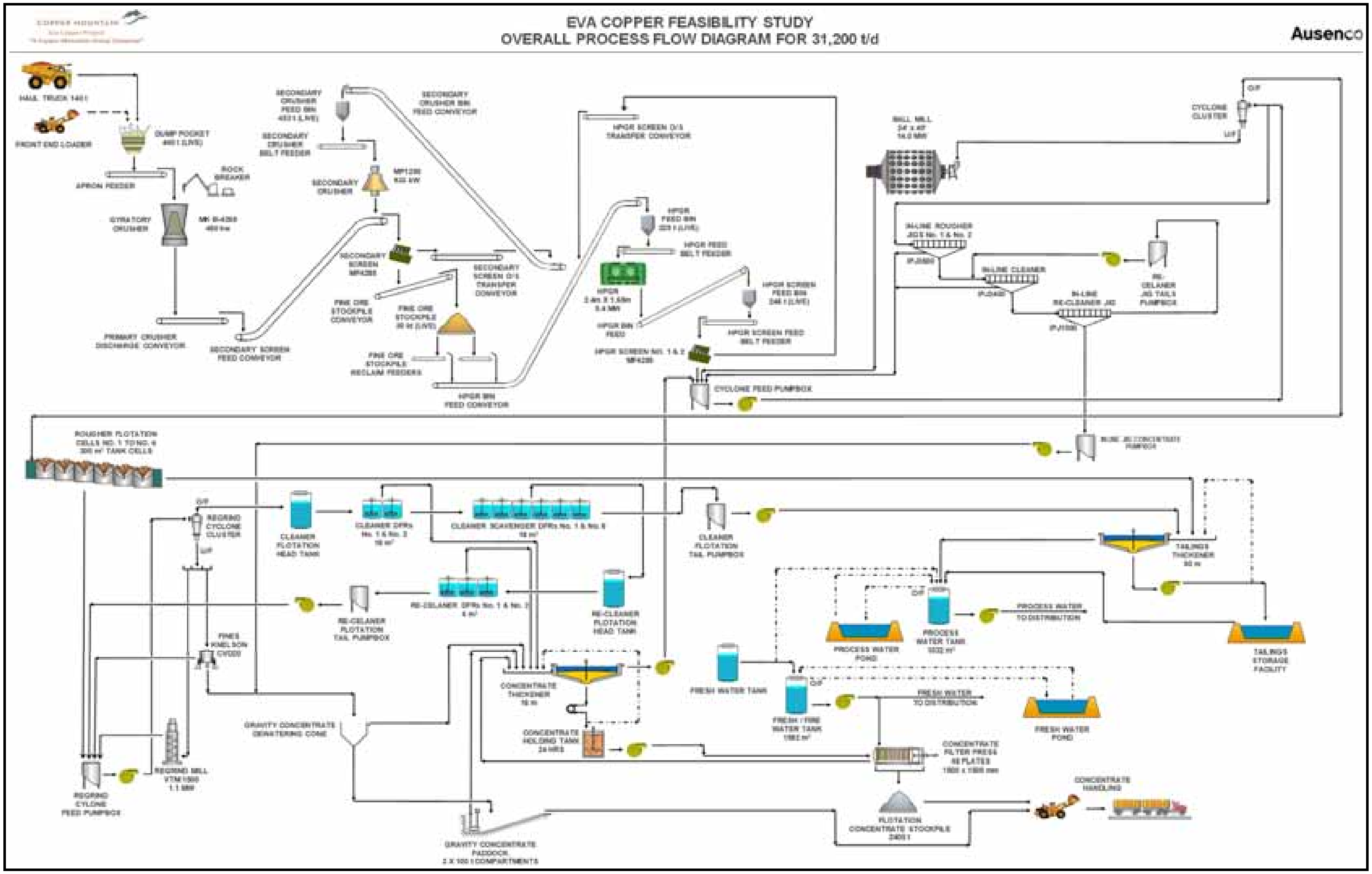


Figure 1-3: Overall Process Flowsheet

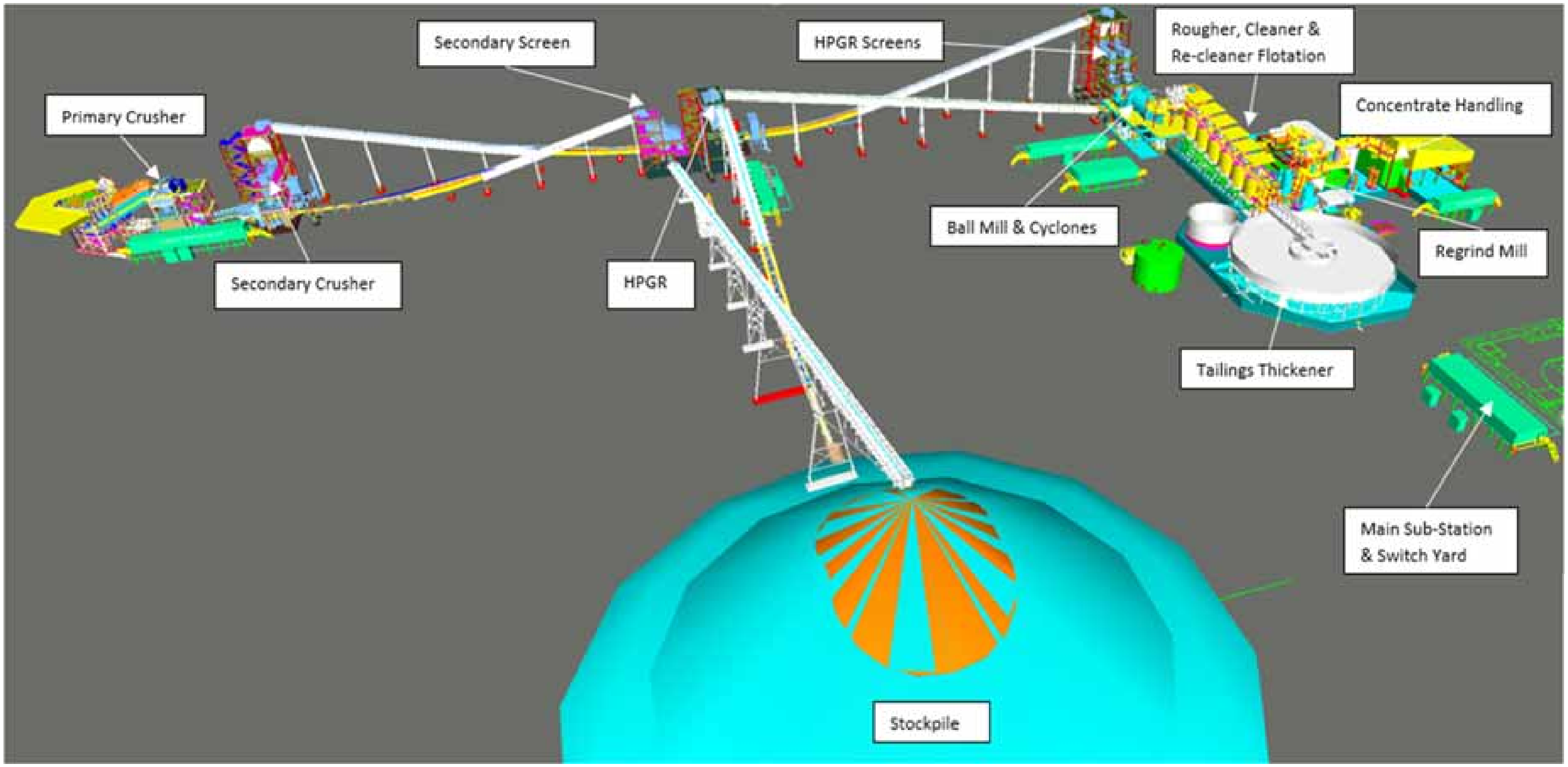


Figure 1-4: Process Plant 3D Layout

1.18 Project Infrastructure

1.18.1 Summary

The Eva Copper mine, plant, and associated open pits are located 76 km northwest of Cloncurry. The site can be accessed by way of the sealed (paved) Burke Development Road, and a planned site access gravel road of approximately 8.5 km.

Infrastructure required to be installed to support the operation includes:

- Roads: main access road, plant site, TSF light vehicle track, explosives and emulsion access road, Cabbage Tree Creek (borefield light vehicle track), and haul roads
- Security office and tag in/out board building
- Administration building, training, first aid, plant crib, and car park
- Control room (primary crusher and rock breaker)
- Control room (grinding area)
- Process plant office
- Concentrate storage shed and weighbridge
- Gravity concentrate paddock
- Reagent storage and building
- Assay laboratory and sample preparation area
- Communication facilities
- Mining infrastructure
- Mine change house
- Truck shop, plant workshop, warehouse, and office
- Tire services pad and services area
- Lubricant storage
- Hydraulic hoses storage
- Fuel storage and dispensing
- Borefields (Little Eva pits and Blackard dewatering wells and Cabbage Tree Creek supply)
- Overland HV transmission line from the tap near Dugald substation (11 km)
- Fresh water supply and treatment
- TSF (424 ha)
- Site sediment management installations
- Creek diversion channel around Little Eva and other pits and surface water bunding
- Explosive bulk storage depot and magazine
- Emulsion facility
- Accommodation village and associated infrastructure.

The broader site infrastructure layout is illustrated in Figure 1-5. For a detailed map of the immediate Little Eva pit and plant area refer to Figure 1-6.

1.18.2 Power Supply

The plant and infrastructure electrical system will be designed and installed to comply with all relevant standards and statutory requirements to provide high reliability and ease of maintenance in accordance with Queensland standards. With 42.5 MW of installed drives, the average power draw for the processing plant during operations will be approximately 30.7 MW.

Power for the processing plant will be supplied from gas-fired generators in Mount Isa for the first three years, at either the Mica Creek power station or the Diamantina power station. Gas supply to these stations is provided by the Carpentaria Gas Pipeline. Power is transmitted along the North West Power System (NWPS) 120 km to the network operator's 220 kV Chumvale substation, adjacent to the town of Cloncurry. From Chumvale, the power is transmitted along MMG's 64 km long, 220 kV, Dugald River overhead transmission line, terminating at MMG's Dugald River substation. A tap will be installed adjacent to the MMG site, and an 11 km extension will be constructed to supply power to the step-down substation (220 kV to 11 kV) at the Project plant site, from which power will be distributed throughout the process plant and to site infrastructure.

The Project has a commercial understanding for access on the MMG Dugald River 220 kV line at the Eva Copper Project demarcation tap point.

For this study the cost of power at site will be US$0.1211/kWh (AU$0.1877/kWh) for the first three years of plant production, based on power transmission from Mount Isa. From year four onwards the cost of power will be US$0.0635/kWh (AU$0.0985/kWh) based on a term sheet with CopperString, the proponent of developing a high voltage electricity transmission line to connect electricity users in the North West Minerals Province (NWMP) and the Mount Isa region to the National Electricity Market (NEM) at Woodstock near Townsville. Figure 1-6 illustrates the layout for site infrastructure.

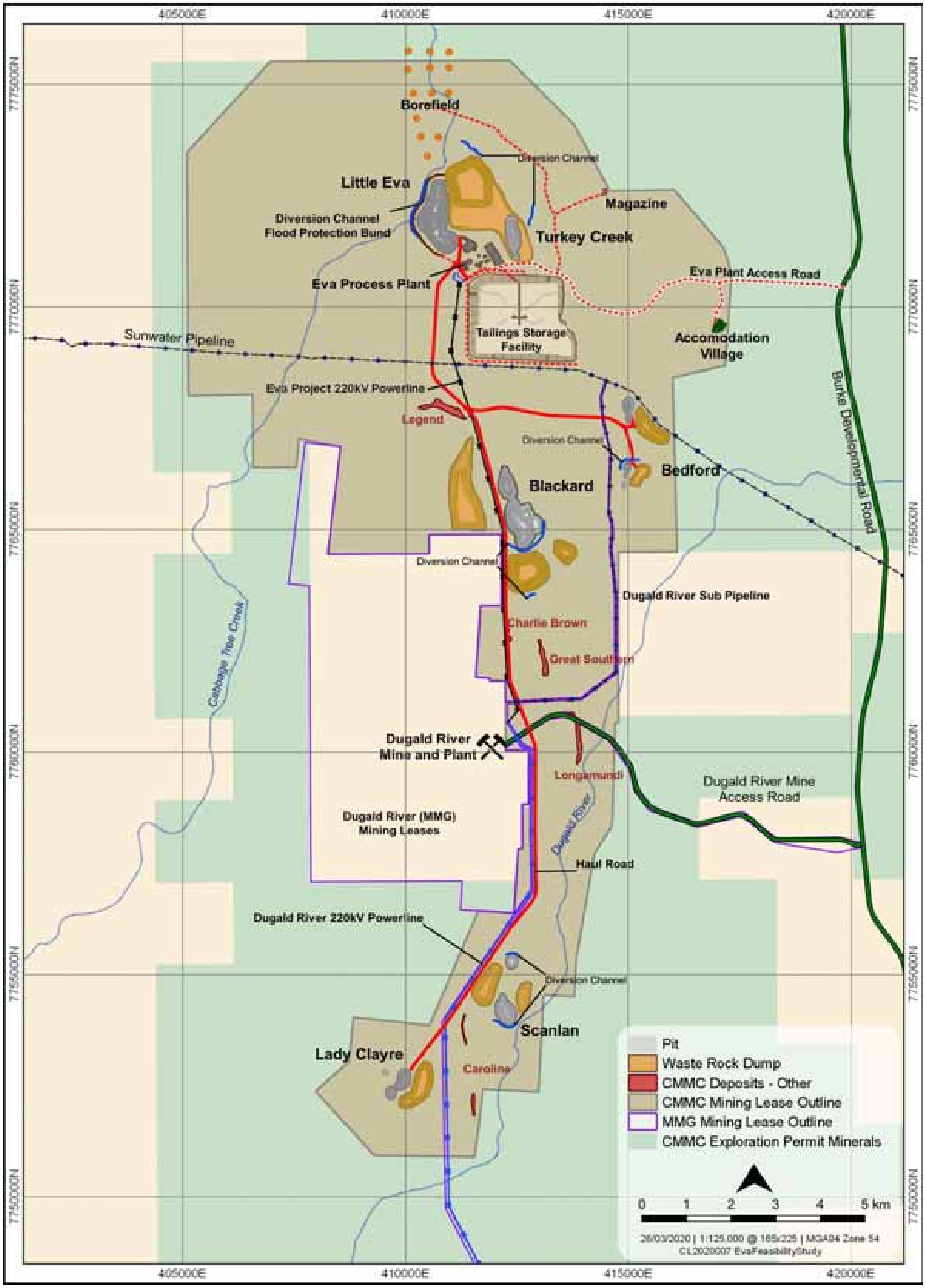


Figure 1-5: Infrastructure and CMMC Tenure

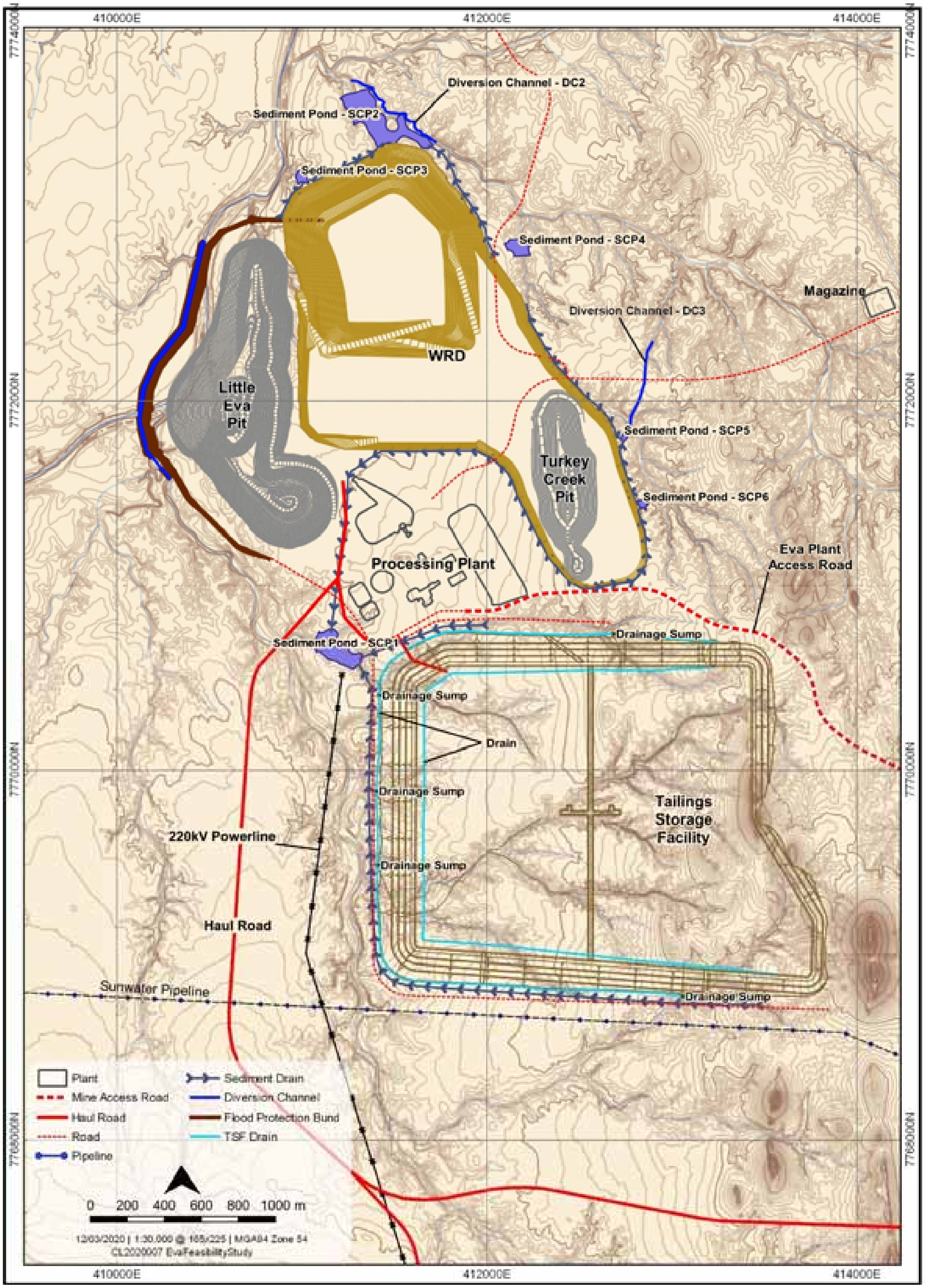


Figure 1-6: Process Plant Area Infrastructure

1.18.3 Water Supply and Treatment

The water requirement for the 11.388 Mt/a (31,200 t/d) processing plant and infrastructure can be supplied by the regional groundwater storages. The Project's water demand is approximately 19,000 m^3/d of which the process plant is approximately 14,000 m^3/d. The water wells will be powered by an 11 kV distribution lines. Supply available is 20,000 m^3/d under average climatic conditions based on the following supply arrangement.

1.18.3.1 Northern Borefield (Cabbage Tree Creek)

This required water will be supplied from a 15-well bore field, 6.5 L/sec per well, at Cabbage Tree Creek located 2 km north of the Little Eva pit at a supply rate of 8,400 m^3/d. The water will be pumped into a 1,200 m^3 nominal capacity northern bores collection tank at the western side of the Little Eva pit on elevation, which will then be transferred to the fresh water/firewater tank at the plant. Two water wells are already cased. Step-rate and constant-rate pumping tests indicated 12 L/sec and 10 L/sec, respectively, sustainable over five years.

1.18.3.2 Little Eva Pit

Little Eva pit dewatering will be accomplished by approximately ten dewatering wells. It is calculated that the ten dewatering wells at 5 L/sec will produce a total of 4,000 m^3/d of raw water supply. The wells will pump through a dedicated pipeline into the 1,200 m^3 nominal capacity northern bores collection tank at the western side of the Little Eva pit.

1.18.3.3 Southern Borefield (Blackard Pit)

This water will be supplied from in and around the Blackard open pit. A 7-well bore field supplying at 3.3 L/sec per well. The southern bore field water will be collected in a 1,200 m^3 nominal capacity collection tank on the eastern side of the Blackard pit and pumped over a distance of 7 km to the processing plant fresh-water tank, at a supply rate of 2,000 m^3/d.

1.18.3.4 Return Tailings Storage Facility Water

Will supply 2,356 m^3/d based on a minimum of 13% return during dry season. The reclaim water will be transferred by a 2-km long HDPE pipeline to the process water tank. During an average wet-season (January to February) up to 80% of return water is available for total of 14,082 m^3/d returned. The average decant-return rate over the course of each year (under average climatic conditions) is 27%, for a total of 4,740 m^3/d.

Moisture content of the ore is estimated to be approximately 3%. Yearly roads dust suppression will amount to approximately 500,000 m^3.

It will also be possible to source additional water from the Lake Julius to SunWater's Ernest Henry pipeline, which is 2 km to the south of the processing plant site.

The potable water for the accommodation village will be supplied from a water well within 1 km from the camp to a tank and then to a water treatment plant to supply 300 L/d per person. A standard reverse osmosis (RO) microfiltration water treatment with UV and/or Chlorine back end dose will be used to ensure potable/drinking water quality.

1.18.4 Tailings Management

KCB was engaged to update a previous TSF Feasibility Study completed by Knight Piésold in 2018. The updated TSF will be located directly south of the processing plant and is classified as a 'High A' consequence facility following ANCOLD (2019) guidelines.

The Eva TSF will be a two-cell paddock facility designed to contain 170 Mt of tailings over approximately 15 years of mine life. The East and West Cells can be operated independently and are separated by a rockfill centre wall, positioned to create cells of approximately equal area. At the ultimate embankment height (maximum ~52 m), the West Cell will have a total footprint area of 216 ha and the East Cell 208 ha. This is a total disturbed footprint area of 424 ha for the TSF.

The perimeter embankments will be zoned earth and rockfill, raised with both the downstream and centreline construction method. The typical dam section will have a sloping upstream low-permeability core, a filter drain, and downstream fine and coarse fill. The filter drain is to control the phreatic surface and provide internal erosion protection, as the decant pond is expected to be against the embankment during the early years of operation.

The TSF will be raised in 12 construction stages. The starter dam (Stage 1) will have a maximum height of approximately 19 m along the western wall. Stages 2 and 3 will be raised by the downstream construction method. By Stages 4 to 12, the decant pond will be centrally located away from the perimeter embankments enabling centreline construction. The filter drain will no longer be required for these raises.

To reduce seepage from the TSF, the impoundment will be lined with compacted low permeability fill material (a combination of reworked in-situ material and imported Zone A fill from selected borrows). Finger drains beneath the TSF embankments, used to lower the phreatic surface in perimeter walls, will drain to collection sumps for pumping back to the TSF.

Tailings will be discharged into the facility by sub-aerial deposition methods, using a combination of spigots at regularly spaced intervals along the perimeter and central embankments. Supernatant water will be removed from the TSF via submersible pumps located within decant towers. Three decant towers will be needed over the lifetime of the facility. The Stage 1 decant tower is located at the centre of the western perimeter wall. Beyond Stage 1, the design intent is to shift the pond towards the central dividing wall between the two cells, where the water return pumps and infrastructure will be located. Solution recovered from the decant system will be pumped back to the processing plant site for reuse in the process circuits.

Seepage and stability analyses have been completed for the LOM and intermediate configurations of the facility. Stability analyses indicate that under static, seismic, and long-term conditions, the TSF meets ANCOLD (2019) design criteria consistent with a High A consequence category facility. This design is currently in the Feasibility Stage and will require additional field investigations and studies during detailed design and prior to construction of the starter embankments.

1.18.5 Logistics

The highway from Cloncurry to Burketown and Normanton on the Gulf of Carpentaria is an existing full-width sealed road that passes 8.5 km to the east of the proposed processing plant site. At Cloncurry, 76 km to the south, it meets the Barkly Highway from Townsville to Mount Isa. Cloncurry has a regional airport, hospital, schools, and other infrastructure.

Concentrate will be trucked to the Glencore smelter at Mount Isa for processing. CMMC has a five-year offtake agreement with Glencore.

Alternatives are available through Townsville port as it is a well-established international port capable of handling bulk mineral materials with over four million tonnes of import/export trade mineral handled annually. All infrastructure required to operate in this manner is already in place and available to the Project.

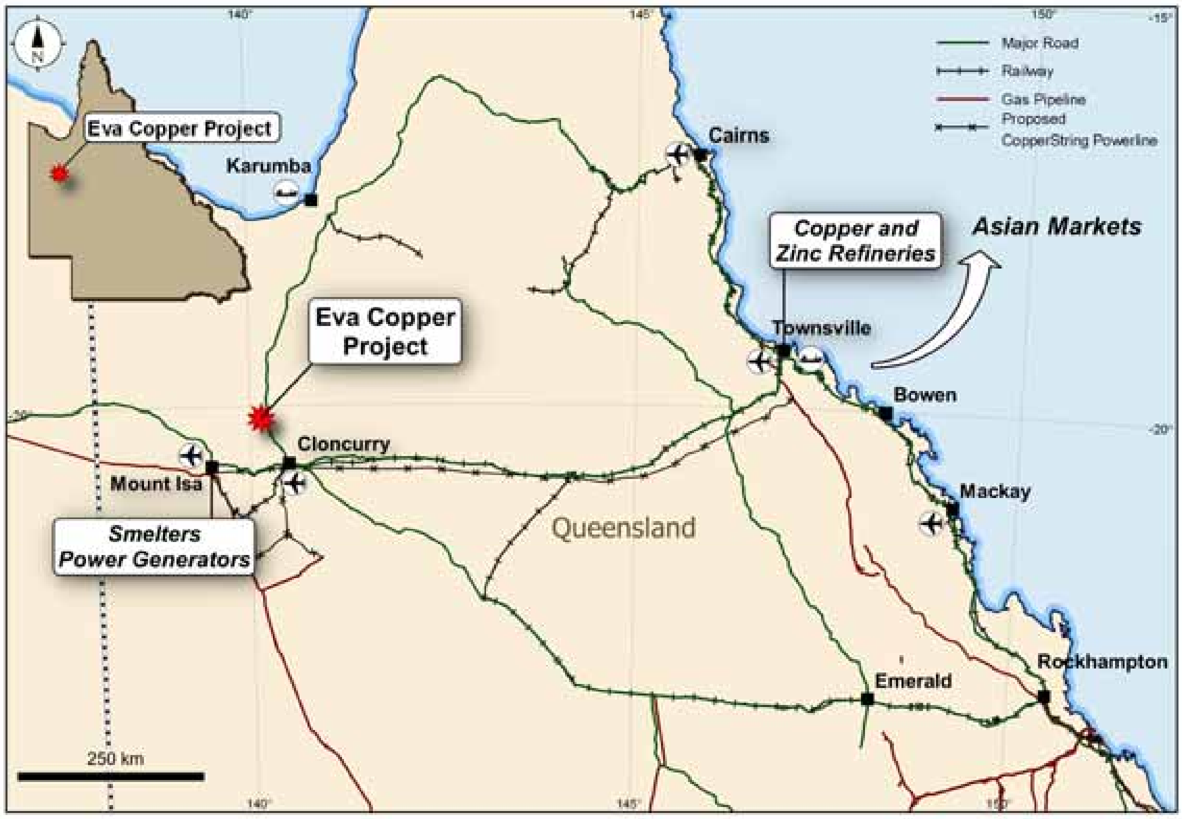


Figure 1-7: Infrastructure in North West Queensland

1.19 Market Studies and Contracts

1.19.1 Concentrate Marketing

The Eva Copper Project will produce a copper concentrate with a LOM grade averaging 28% Cu and 3 g/dmt Au. The mine is expected to produce on average 163,000 dmt/a of copper concentrate over the LOM. The material will be considered a “clean concentrate” with no deleterious elements that would cause smelters to penalize the material.

An offtake agreement has been finalized, with Glencore International AG for a hundred percent (100%) of the mine’s output, with a fixed duration of five years and commencing with the start of mine production. The contract may be extended for a further five-year period, by mutual-agreement. The sale of the concentrate will be made on basis as freight carrier at (FCA) Seller’s mine gate.

Treatment and refining charges with fees paid to smelters by mines for converting the concentrate into refined copper, will be based on the annual prevailing market terms (annual benchmark) established between major international copper concentrate producers and major Japanese smelting companies. These charges will reflect current market fundamentals at the time of sale.

Discussions with other potential off-takers (smelting companies and concentrate trading companies) indicated interest in Eva concentrates should they become available at the end of the initial offtake agreement. The marketing cost assumptions are based on discussions with major smelters and concentrate trading companies and on the Company's own views and experience in the copper concentrate market.

1.19.2 Copper Price Forecast

The lack of investment in copper mines and mine expansions lead many analysts to believe that there will be a tighter market for copper concentrates well into the 2020s. On the other hand, forecasted world copper demand, fuelled by electronic vehicles and renewable energy, is expected to see growth well into the future. The increase in demand and the lack of commitment on the supply side tends to give support to the copper price (Table 1-7).

Table 1-7: Copper Prices

	2020	2021	2022	2023	Long Term
Copper Prices ($/lb)	2.84	2.89	2.97	3.03	3.04

Source: CIBC Global Mining Group – Consensus Commodity Price Forecasts February 28, 2020.

1.19.3 Smelter Charges

Copper concentrates are sold by mines to smelting companies and merchants who charge treatment and refining charges (TC/RCs) to process the material. TC/RCs increase in an over supplied market and decrease when concentrate availability is tight. Treatment charges are calculated per dry tonne (dmt) of concentrate and refining charges are calculated per pound of payable copper. Consensus points to a tight concentrate market given the limited project development as well as expected smelter expansion required to meet the copper demand. This is especially true in China where deficits are forecasted for the next several years.

Table 1-8: Smelter Charges

	2020 Benchmark	2021 Forecast	2022 Forecast	2023 Forecast	Long Term Forecast
Treatment Charges ($/dmt)	62.00	75.00	75.00	75.00	76.00
Refining Charges (¢/lb)	6.20	7.50	7.50	7.50	7.60
Total TC/RC (¢/lb)	**16.62**	**20.10**	**20.10**	**20.10**	**20.37**

Other terms used in the study are internationally recognized standards for copper and precious metal payables and precious metal refining charges.

- Copper: 96.5% with a minimum 1-unit deduction
- Gold: 92.0% with gold content between 3 g/t and 5 g/t; and
  94.0% with gold content between 5 g/t and 7 g/t

The typical refining charge for gold at this grade range is $5/oz.

1.19.4 Precious Metal Prices

Table 1-9 shows the precious metal prices obtain from CIBC Global Mining Group—Consensus Commodity Price Forecasts February 28, 2020.

Table 1-9: Precious Metal Prices

	2020	2021	2022	2023	Long Term
Gold ($/oz)	1,521	1,507	1,466	1,434	1,362

1.19.5 Concentrate Markets

With a long-term off-take agreement now in place Eva copper concentrates are fully committed for the first five years of production; however, if the contract is not extended past the present agreement, other markets would be readily available.

The copper concentrate market is predicted to move to a deficit position in the next few years as global copper concentrate output is expected to grow at a slower rate, making it difficult to meet demand of expanded smelting capacities. China is expected to continue to expand its smelting capacity and although there are no firm smelter projects outside China, additional smelter capacity in countries such as Indonesia, Iran, Mongolia, and Zambia, are strong candidates for potential recipients of Eva Copper Project concentrates. Governments in developing economies that have mine production are also looking for additional concentrates to ensure enough smelting capacity to treat concentrates locally.

Should the initial sale and purchase agreement not be extended, the clean concentrates produced at the Eva mine would have no trouble finding a home in Asian smelters or with international trading companies.

1.19.6 Royalties

State of Queensland royalties apply to all lands except freehold claims prior to 1904. State royalties range between 2.5% and 5.0% of metal value, less certain allowable expenses. If the concentrate is processed in Queensland (Mount Isa) there is a 20% reduction in the copper royalty. 100% of the royalty savings from the Queensland Government is for the account of the Seller (CMMC). Royalties are discussed in detail in Section 4.

1.20 Environment, Permitting, Social, or Community Impact

To support EA applications, flora and fauna surveys and waste and tailings rock characterization were undertaken. This characterization work was also done to support mining of the open pits, location of the waste dump, TSF, and Cabbage Tree Creek diversion bund and channel. From flora and fauna surveys the key management issues relates to three regional ecosystems listed as “endangered” or “of concern” that generally have a restricted distribution along major drainages. Clearing in these areas triggers an environmental offset requirement (in the form of a financial settlement or conservation work programs to be approved by DES).

The Project area is uninhabited with the closest sensitive receptor being Mount Roseby homestead, approximately 17.5 km southeast of Little Eva pit and processing plant while the closest pit, Scanlan is 1.1 km west of Mount Roseby. Noise and air quality monitoring is a requirement of the EA, and dust baseline monitoring has been completed.

Tailings and waste characterization work has shown both to be geochemically benign.

As a condition of the EA, water and sediment management requires surface water and groundwater monitoring programs prior to commencement of mining activities. Baseline water and sediment quality monitoring programs have been in place since 2012 and were expanded in 2018 with new baseline monitoring wells established at Little Eva Turkey Creek, the TSF, and the processing plant location as required ahead of mine construction.

The key risks associated with release of contaminants into the environment have been considered with the TSF, waste rock dump (WRD), and processing plant area designs incorporating surface water management control dams, cut-off drains, monitoring, and low permeability basin for the TSF. Waste dumps will be rehabilitated to ensure revegetation of the area.

The evidence of European history in the area is not of local or State significance. The recognized traditional owners and Native Title holders of the Project area are the Kalkadoon People. The Company has a Cultural Heritage and Access Agreement and Management Plan with the Native Title holders covering the full area of the Project MLs. The ML area has been the subject of systematic Indigenous cultural heritage clearing, protection, and management programs.

In addition to managing environmental and heritage responsibilities the Company recognizes and has reflected the importance it places on building and training its workforce, supporting the local community and stakeholders, and a commitment to achieve the highest standards of safety and health for its business practices. While the operation will be dependent on FIFO and drive in/drive out employees, the Company is committed to employing residents from the community and senior employees in professional and technical roles will be offered the option of relocating to Cloncurry at the Company’s expense. Through our agreement with the Kalkadoon People, the Company will strive to provide employment opportunities for local Indigenous people. The key community risk requiring management from commencement of operations through the LOM will be the additional vehicular traffic along the Burke Developmental Road and through Cloncurry.

1.21 Capital and Operating Costs

1.21.1 Capital Costs

The capital cost estimate for the Project was developed by Merit Consultants International Inc. A Division of Cementation Canada Inc. with input from CMMC and various independent engineers and consultants according to their scope of work.

The capital cost estimate is based on a combination of equipment supplier quotes, supplier pricing, construction contractor input, and experience with similar sized operations. This Project estimate meets the American Association of Cost Engineers (AACE) Class 3 requirements and is prepared to form the basis for budget authorization, appropriation, and/or funding purposes. It has an expected accuracy range of ±15%.

This capital cost estimate assumes contracts will be awarded to reputable contractors on a lump sum or unit price basis in an open shop environment.

The CAPEX was prepared in Australian dollars which were converted to United States dollars using a rate of 1.55 at the time of preparation in Q1 2020. The projected initial modelled development capital cost for the Project is estimated at US$454.5 million, including a US$41.5 million contingency allocation (equates to 10% of the direct and indirect costs). There are approximately 1.6 million direct and indirect man-hours associated with the construction of the Project, including pre-production personnel with the workforce peaking at 450 people. Estimated capital costs are shown in Table 1-10.

Table 1-10: Eva Copper Project Development Capital Cost Summary

Capital Cost Items	Initial Years (Year -2 to Year 1) (US$ millions)	Year 2 to Year 15 (US$ millions)	Total CAPEX (US$ millions)
Direct Costs			
Mining	35.2	61.4	96.6
Process Plant	150.8	-	150.8
Infrastructure	67.6	-	67.6
Ancillaries	25.6	-	25.6
Total Direct Costs	**279.3**	**-**	**340.6**
Indirect Costs			
EPCM	25.1	-	25.1
Freight and Logistics	7.6	-	7.6
Indirect Costs	24.3	-	24.3
Owner's Costs	15.3	-	15.3
Total Indirect Costs	**72.3**	**-**	**72.3**
Subtotal	351.5	61.4	412.9
Contingency	41.5	-	41.5
Total Project Capital	**393.1**	**-**	**454.5**[(1)]
Pre-production revenues	(11.2)	-	(11.2)

Capital Cost Items	Initial Years (Year -2 to Year 1) (US$ millions)	Year 2 to Year 15 (US$ millions)	Total CAPEX (US$ millions)
Total Capital	**382.0**	**61.4**	**443.4**
Sustaining capital	-	34.0	34.0
Rehabilitation	1.28	12.9	14.1
Overall Project Capital	383.3	108.2	491.5

Note: (1)Total Project CAPEX is 704.5 in Australian dollars.

1.21.2 Operating Costs

Operating cost estimates as shown in Table 1-11 are based on Aon plc's McDonald Gold and General Mining Industries Remuneration Report Australasia Q2 2018 labour rates and supply quotations direct from suppliers. Mining rates are based on the Company assuming the performance of mining activities. Quotations for explosives and mining consumable supplies are based on Q4 2019 supplier bids and fully support cost model build-up.

Power costs are based on indicative energy term sheets and invoice summary received in Q4 2019 The term sheets includes pricing based on contracted energy capacity and an indicative supply arrangement for natural gas supply and transport arrangement. It also includes costs for renewable energy target Large Generation Certificates (LGC) and Small-Scale Technology Certificates (STC), including the CopperString term sheet from year four onwards.

Table 1-11: Operating Cost Estimate – Summary by Area

Operating Cost Area	LOM Total ($ million)	Unit Cost ($/t milled)
Mining	888.7	5.26
Processing	868.3	5.14
G&A	95.0	0.56
Accommodation and Travel	72.4	0.43
Total	**1,924.5**	**11.39**

Notes: Total mining costs are estimated at $5.26/t milled, or $1.66/t mined. Royalties for LOM total is $199.9 million at a unit cost of $1.18/t milled.

1.22 Economic Analysis

An economic model was developed to reflect projected annual cash flows and sensitivities of the Project. The economic model was created using various assumptions that are based on current and projected economic conditions including, but not limited to, sales prices, operating costs, annual production, ore grades, and exchange rates. All costs, metal prices and economic results are reported in United States dollars ($) unless otherwise stated.

The Key Inputs and Assumptions used are outlined in Table 1-12.

Table 1-12: Eva Project – Key Inputs and Assumptions (Average LOM Values)

Parameter	Unit	Value
Mine Life	years	15
Total Ore	Mt	170
Total Waste (including 14,074 kt of oxide material)	Mt	381
Processing Rate	t/d	31,200
Average Cu Head Grade	%	0.46
Cu Recoveries	%	87
Au Recoveries	%	78
Cu Produced	Mlb	1,502
Au Produced	koz	205
Cu Price (long-term from Year 2)	US$/lb	3.04
Au Price (long-term from Year 2)	US$/oz	1,362
Exchange Rate (long-term)	AU$:US$	1.55

Other key inputs and economic factors include the following:

- Discount rate of 8% (sensitivities of other discount rates have been calculated).
- Revenues, costs, and taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment.
- Progressive reclamation totalling to $14 million over the LOM.
- Nominal 2020 dollars with no inflation and on a constant dollar basis.
- Results are presented on a 100% basis; do not include management fees. The Capital cost of $49.6 million of mine equipment purchased in Year -1 and Year 1 has been amortized over a lease term of seven years at 5%.
- All pre-development and sunk costs, such as exploration and resource definition costs, engineering fieldwork and studies costs, and environmental baseline studies, were excluded. However, pre-development and sunk costs are utilized in the tax calculations.

Table 1-13 presents a recent update on currencies based on consensus views of major Canadian and Australian banks taking into consideration the current economic volatility.

Table 1-13: Commonwealth Bank of Australia, March 31, 2020 (End Quarter Forecast)

	Mar-20	Jun-20	Sep-20	Dec-20	Mar-21
AU$:US$	0.6000	0.5700	0.6000	0.6300	0.6500
Old Forecast	0.6800	0.6700	0.6800	0.6800	0.6700
Forward Market	0.6140	0.6136	0.6131	0.6124	0.6118
Consensus	0.6700	0.6700	0.6800	0.6800	0.6800

Source: CBA, Bloomberg

The Project is economically viable with an after-tax IRR of 29% and NPV at 8% of 437 million. Figure 1-8 shows the projected cash flows from the economic analysis and Table 1-14 summarizes the detailed results of this evaluation.

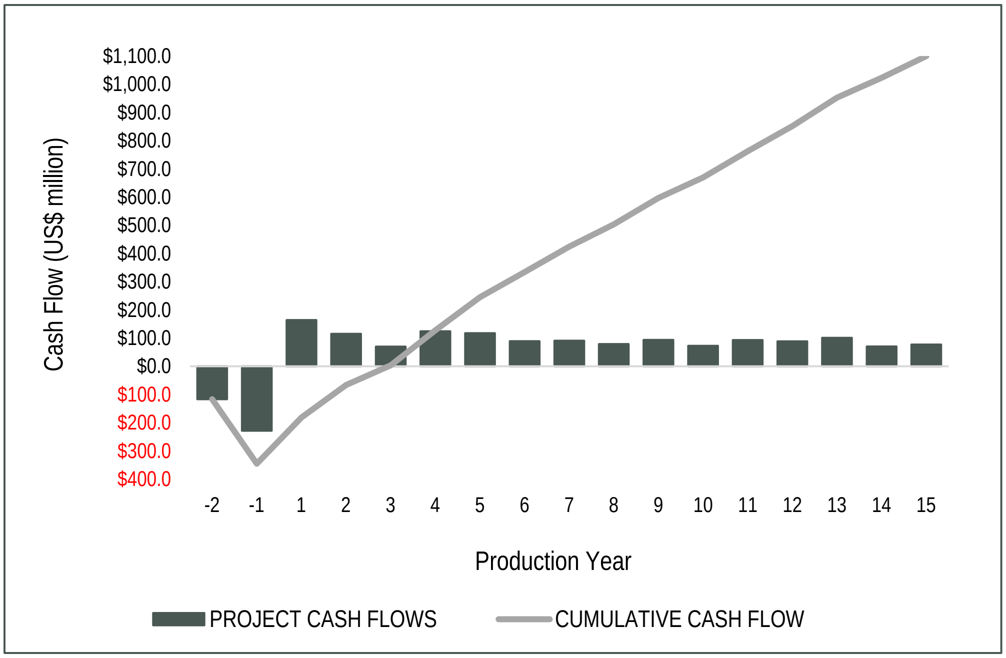


Figure 1-8: Eva Copper Project Annual and Cumulative After-Tax Cash Flows

Table 1-14: Summary of Economic Results

Key Financial Metrics	Unit	Value
Net Revenues	$ million	4,311
Operating Costs	$ million	1,924
Cash Flow from Operations	$ million	2,387
Royalties and Transportation	$ million	371
Taxes	$ million	447
Cash Flow after Taxes	$ million	1,568
Sustaining Capital Costs	$ million	34
Cash Flow after Taxes and Sustaining Capital	$ million	1,534
C1 Cash Cost per Pound of Copper Produced After Credits	$ million	1.44
Cash Cost per Pound Produced (after taxes and sustaining capital)	$ million	1.76
Pre-Tax NPV 8%	$ million	648
Pre-Tax IRR	%	37
After-Tax NPV 8%	$ million	437
After-Tax IRR	%	29

1.23 Adjacent Properties

The Eva Copper Project is located within a world-class mineral province richly endowed with an attractive number of commodities and deposit types. It is commonly known that the Mount Isa – Cloncurry region is one of the premier base-metal producing districts in the world with mining dating back to 1867, first at Cloncurry, then from the larger Mount Isa mining centre starting in 1923. There are numerous historical and active mines in the region, with the major, internationally important mines closest to the Project being the Dugald River lead-zinc-silver mine and the Ernest Henry copper-gold mine. Dugald River is the closest, located approximately 11 km south of the proposed Eva Copper Project processing plant site.

Mining properties that surround the Eva Copper Project are predominantly Exploration Permits for Minerals held by CMMC. These permits cover a highly prospective north-south corridor, with similar geology to that which hosts the Project's Mineral Resources. Numerous copper-gold mineralized prospects have been established and are being systematically explored.

Immediate non-mining key local stakeholders associated with the Eva Copper Project are landowners, leaseholders, state, and local governments. The Company has been in contact with the stakeholders for many years and has appropriate agreements in place to allow mining and exploration.

1.24 Other Relevant Data and Information

1.24.1 Project Execution Plan Outline

The Eva Copper deposits will be mined at a rate to produce approximately 31,200 t/d ROM ore for direct feed to the process plant. The flowsheet developed for the Project is a relatively standard copper sulphide processing plant. All the unit operations used are low-risk, proven technology.

- The design life is 15 years
- Key process units include:
    - Single Stage Gyratory Crushing (dual dump capability)
    - Secondary Crushing with closed circuit screen
    - HPGR Tertiary Crushing in closed circuit with wet screening undersize to ball milling
    - Gravity Jig Concentration as a bleed of the cyclone feed line
    - Flotation (Conventional Rougher and DFR Cleaner Scavenger)
    - Concentrate Regrind (Vertimill®), one Cleaner stage and one Recleaner stage
    - Gravity bowl concentrator off regrind cyclone underflow
    - Concentrate Thickening and Filtration
    - Tailings Thickening and disposal.

The Project Execution Plan (PEP) outline has been prepared for the Eva Copper Project's updated Feasibility Study. The Project Execution Plan is intended to define all the base activities of engineering, procurement, construction, and environmental activities, and to ensure that the core elements of the Eva Copper Project's sustainable development framework regarding the inter-relationship between the stakeholders, including pastoralists, community members and Native Title holders, and project development are maintained throughout.

CMMC's long- and short-term objectives and integrated assessment of the dimensions of sustainable development encompass the entire mineral exploration to production chain. The core elements of the Eva Copper Project's sustainable development framework are as discussed below.

1.24.1.1 Human Rights

The Company is committed to uphold fundamental human rights and respect cultures, customs, and values in dealing with communities, employees, and others affected by the Company's activities.

1.24.1.2 Project Due-Diligence and Pre-Engagement

The Company is committed to remain informed of the political, economic, social, technical, and environmental characteristics of the area in which it operates. Sound data obtained will contribute to the design and structure of risk management strategies, as well as pre-engagement processes such as preparation for field activities.

1.24.1.3 Community and Aboriginal Engagement and Enhancement

The Company is committed to develop long-lasting economic, environmental, and social benefits through the building of meaningful and transparent relationships with local communities and Native Title holders.

1.24.1.4 Human Resource Development

The Company is committed to provide long-term benefits for the community through areas, such as employment, training, and education.

1.24.1.5 Environmental Integrity and Performance

The Company is committed to manage all operations in a manner that is compatible with environmental protection standards and integrate closure requirements into all stages of the Company's activities.

1.24.1.6 Health and Safety Performance

The Company is committed to provide a safe environment for employees, contractors, and visitors to the Company's facilities, and a commitment to support leadership in preventive and responsive attitudes and behaviours at all levels of the organization to ensure a safe environment.

1.24.2 Engineering, Procurement, and Construction Management

Under the administration of the Owner's Project Manager, the engineering, procurement, and construction (EPCM) team, which will consist of a combination of Owner's personnel embedded within a contracted CM firm, will manage the Project in accordance with the Project schedule, capital cost, health and safety, environmental, and quality targets.

Once the Execution Plan has been finalized and implemented, it is important not to deviate from its original intent, as variances may translate to a change in a specific project driver. Modifications made to the plan should only be done with the intent of improving the base elements of the Execution Plan, without compromising the established sustainable development framework, which takes a disciplined and integrated approach to the Company's activities in areas of governance, social development, economic contributions, and environmental stewardship.

1.24.3 Project Execution Schedule

Merit has prepared a feasibility level schedule that will become the baseline schedule. The overall Project schedule identifies the preferred critical sequences and target milestone dates that need to be managed for the Project to be executed successfully. The future detailed schedules will track the planned and actual progress throughout the duration of the Project using information provided by the engineering groups, contractors, suppliers, field management staff and CMMC.

The total duration for the Project completion is estimated to be 29 months, from start of early infrastructure engineering to commissioning complete. This includes a 19 month project construction duration and assumes commencement of field activities in Month -22 and mechanical completion in Month -3. Detailed engineering is scheduled to start in Month -22 to allow sufficient progress to award mostly fixed-price construction contracts. The purchase of major process equipment is assumed to be completed by Month -8. The schedule accounts for Christmas breaks and rainy season "rain-out" days."

The Feasibility Study Project schedule reflects the EA approval timeline and permits required to be in place to enable commencement of construction activities in Month -22. Detailed engineering is expected to achieve substantial completion in Month -12. A 220 kV powerline and a 220 kV / 11 kV main substation will be ready to energize by the utility by Month -9, allowing for some pre-commissioning activities to start as soon as they are able to and also allow the accommodation village to come off generator power as early as possible. The Key Project Milestones are shown in Table 1-15.

Table 1-15: Key Project Milestones

Milestone	Date
Early Infrastructure Engineering Starts	-29 months
Project Approval and Start	-25 months
Basic Engineering Complete	-22 months
Detail Engineering Complete	-12 months
Full Construction Starts	-22 months
Utility Power Required	-9 months
Tailings Storage Facility Complete	-3 months
Mechanical Completion	-3 months
Hot Commissioning Starts	-3 months
Commercial Production Starts	Month 1

1.25 Interpretation and Conclusions

The Project area is uninhabited, with the closest sensitive receptor being Mount Roseby Homestead, which is approximately 17.5 km southeast of the Little Eva pit and processing plant and 1.1 km from the Scanlan pit. Noise and air quality monitoring is a requirement of the EA.

1.25.1 Geology, Mineral Resources, and Mineral Reserves

- The Eva Copper Project Mineral Resources are IOCG deposits that vary according to setting. The main deposit, Little Eva, is similar to Ernest Henry.
- Mineralization primarily occurs as chalcopyrite.
- The mineralized zones typically trend north to south and are moderate to steeply dipping.
- The Mineral Reserves listed in Table 15-1 comply with all disclosure requirements for Mineral Resources set out in NI 43-101.
- CMMC and Stuart Collins, P.E., believe the Mineral Reserves are being estimated in an appropriate manner using current mining software and procedures consistent with reasonable practices. The classification of Measured, Indicated, and Inferred Resources conform to Canadian Institute of Mining, Metallurgy, and Petroleum Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM Definition Standards).
- Mr. Collins is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that would materially affect the Mineral Reserve estimates.

1.25.2 Mining

- Conventional open pit mining methods (drilling, blasting, loading, hauling) will be employed to extract the ore and waste.
- There are seven deposits to be mined: Little Eva, Turkey Creek, Bedford, Lady Clayre, Ivy Ann, Blackard, and Scanlan. None of the deposits has previously been mined. Little Eva, Turkey Creek, Bedford, Lady Clayre, and Ivy Ann represent approximately 70% of the Mineral Reserves.
- Mining by CMMC personnel will begin in the Little Eva pit (Year 1), and in the Blackard and Turkey Creek pits from Year 2 onwards. Mine life is 15 years, with a one-year mining preproduction period. The Project's overall strip ratio (waste tonnes to ore tonnes) is 2.2:1.
- The mine plan estimates that there are 170 Mt of ore grading 0.46% Cu and 0.05 g/t Au, and 381 Mt of waste will be generated over the LOM.
- Topographical relief, climate, haul distances, and geographic location present no issues to the Project.
- Factors that could impact production if not addressed by CMMC are dewatering the pit and slope stability.

1.25.3 Metallurgical Testwork and Mineral Processing

- The competency and hardness values for the 75:25 blend of sulphides and native copper ore sources indicates 31,200 t/d at 165 μm grind is achievable with the updated plant design.
- Little Eva, being the largest source of sulphide ore, is expected to see 95% recovery. The remaining sulphide ore sources are expected to see between 88% to 95% depending on the mineralogy.
- Blackard and Scanlan native copper zones are expected to achieve 63% recovery through gravity and flotation recovery methods.
- The recovery within the native copper zone of Blackard will be variable; however, will average to 63% as shown in testwork. The sulphide zone located below this, is expected to behave similar to Turkey Creek at 88% recovery.

- Extensive work has been done on Blackard. Scanlan has not seen the same degree of study; however, pilot flotation work, and geological observations have shown for it to have the same mineralogical characteristics as Blackard.

1.25.4 Process Plant

- The process plant flowsheet is a standard processing plant design featuring a two-stage crushing and HPGR format with a gravity recovery circuit installed on the ball mill and regrind cyclone loops.
- The processing plant has been designed to produce a marketable copper concentrate grade of 28% Cu and about 3 g/t Au.
- The daily average throughput is 31,200 t/d over the mine life based on conservative ore competency and hardness values.

1.25.5 Infrastructure

This greenfield project will require the following major components to be built:

- Access and site roads
- Accommodation village for Project construction and operational personnel
- An 11.44 Mt/a capacity crushing, milling, and flotation process plant
- A 11 km, 220 kVA transmission line, substation, and site distribution electrical system
- A water supply system to provide approximately 19,000 m^3/d of water
- Site administration office complex, and a six-bay truck and plant maintenance shop with attached warehouse facilities
- Tailings storage facility
- Site sediment management installations
- Cabbage Tree Creek diversion channel around the Little Eva pit, and surface water bunding
- Fuel storage and dispensing
- Plant site laboratory
- Communication facilities
- Training and first aid facilities
- Open pit mining infrastructure
- Borefield dewatering wells for the open pits, and the Cabbage Tree Creek supply
- Explosives bulk storage depot and magazine.

1.25.6 Environmental, Permitting, and Social Considerations

- MLs and an EA for the Project have been granted. The EA from the DES regulates the environmental management of the Project and sets out key environmental management conditions. The current EA is based on the previous 2016 mine layout. Changes to the mine layout and throughput increases set out by this Feasibility Study update will require submission of a Major Amendment. These are straightforward procedural processes.

- To support EA applications, all baseline studies (like flora and fauna surveys, or waste and tailings rock characterization) have been undertaken, and these included work to support mining of the open pits, and location of the waste dump, TSF, mine access road, and Cabbage Tree Creek diversion bund and channel.
- The Project area is uninhabited, with the closest sensitive receptor being Mount Roseby Homestead, which is approximately 17.5 km southeast of the Little Eva pit and processing plant and 1.1 km from the Scanlan pit. Noise and air quality monitoring is a requirement of the EA.
- The key risks associated with release of contaminants into the environment have been considered, with the design incorporating surface water management control dams and inclusion in the TSF design of a low-permeability basin, cut-off drains, and monitoring.

1.25.7 Capital and Operating Costs

- Approximately 300 full-time jobs will be directly created by this Project.
- The total Project capital is approximately $454.5 million, and sustaining capital is estimated to be $34.0 million at an assumed exchange rate of AU$1.55 to US$1.
- Average LOM operating costs are estimated to be $11.39/t milled (excluding royalties). The C1 cash cost is estimated at $1.44/lb.

1.25.8 Economics

- The Project has a recoverable copper content of 1,502 Mlb of copper and 205 koz of gold over a 15-year life.
- Project economics are good at a long-term copper price of $3.04/lb and a long-term gold price of $1,362/oz.
- A long-term exchange rate of AUS$1.55 to US$1 was used.
- At a discount rate of 8%, the after-tax NPV is $437 million, and the after-tax IRR is 29%.
- This Project is most sensitive to the copper price, copper recoveries, and copper head grade delivered to the process plant. The exchange rate, operating costs, and capital costs may also impact the Project's economics to a lesser degree.

1.26 Recommendations

1.26.1 Mineral Resources and Mineral Reserves

- Drill targets below and within the current pit designs to convert Inferred Resources to Indicated Resources.
- At the Little Eva pit, conduct development drilling ahead of mining to optimize mining selectivity and grade control costs/strategy.
- Perform geotechnical slope studies on the Turkey Creek, Lady Clayre, Bedford, and Ivy Ann deposits.
- Continue detailed mine design and mine planning on the Eva Copper Project prior to production.
- Develop detailed dewatering plans for the Little Eva, Blackard and Turkey Creek pits.

1.26.2 Infrastructure, Process, and Plant

- Perform confirmatory geotechnical investigation of Cabbage Tree Creek bund and the TSF second cell western side.
- Re-evaluate the hydrology and dewatering of the Little Eva, Blackard and Turkey Creek pits in the context of the new geotechnical models.
- Redo the overall site Hydrogeology Report, last done by KH Morgan and Associates (Morgan) in December 2009, to include the Cabbage Creek borefield and potential water bore source for the accommodation village.
- Perform follow up testwork to investigate further improvement of final grade by means of magnetic separation. Some testwork has highlighted that this is an effective means of removing iron bearing minerals, barren of copper, from final concentrate during coarse gravity separation. This combined with additional investigation into the cleaner circuit could yield further improvement on final product grades, improving the economics of the Project.
- Investigate the potential of the gravity concentrate bypassing the smelting process, that might attract a slightly elevated price per tonne.
- Scanlan ore was studied during bench and pilot tests performed in 2006. There is no recent data on this ore source; however, all data and geological observations indicate equivalent behaviour to Blackard ore. Additional testwork and spatial variability investigations should be performed to enhance the understanding of this deposit, even though the mining plan indicate for Scanlan ore to only start in Year 7. There is no data available on the deeper sulphide portion of this deposit.

1.26.3 Project Environmental Authority (EA EPML00899613)

Both the MLs and the EA have been approved. Changes made to the mine layout in this feasibility study require a new amendment to the exiting EA. Amendments are assessed to determine whether they are classified as Minor or Major. The extent of the new mine footprint, increased processing throughputs, adjustments to the waste dump, plant areas, TSF, Cabbage Tree Creek water well field, and road routes, and inclusion of the Blackard and Scanlan deposits to the mine plan will require submission of a Major Amendment Application to the existing EA. From the date of application submission, the Minor Amendment process takes up to 35 days, while the time for a Major Amendment can vary. The 2016 Major Amendment by Altona took 3.5 months from the date of application submission.

2 INTRODUCTION

Copper Mountain Mining Pty. Ltd. (CMMPL) is a wholly-owned subsidiary of Copper Mountain Mining Corporation (CMMC or the Company) and is located in Queensland, Australia. The Eva Copper Project (the Project) is located approximately 76 kilometres (km) northwest of Cloncurry in North West Queensland, Australia, and has extensive exploration potential in the approximately 4,000 square kilometres (km^2) (379,000 hectare (ha)) mineralized land package.

CMMC commissioned Ausenco Engineering Canada Inc. (Ausenco) to redesign and redevelop the 2018 Feasibility Study process plant and associated site infrastructure, and to provide coordination services and technical input into the preparation of this National Instrument (NI) 43-101-compliant feasibility level technical report. In addition, CMMC commissioned Klohn Crippen Berger (KCB) to redesign the 2018 Knight Piésold Ltd. (Knight Piésold) tailings storage facility (TSF) and to provide input to water management, and Merit Consultants International (Merit), a division of Cementation Canada Inc., to develop the capital cost, construction management, and execution plan of the Project.

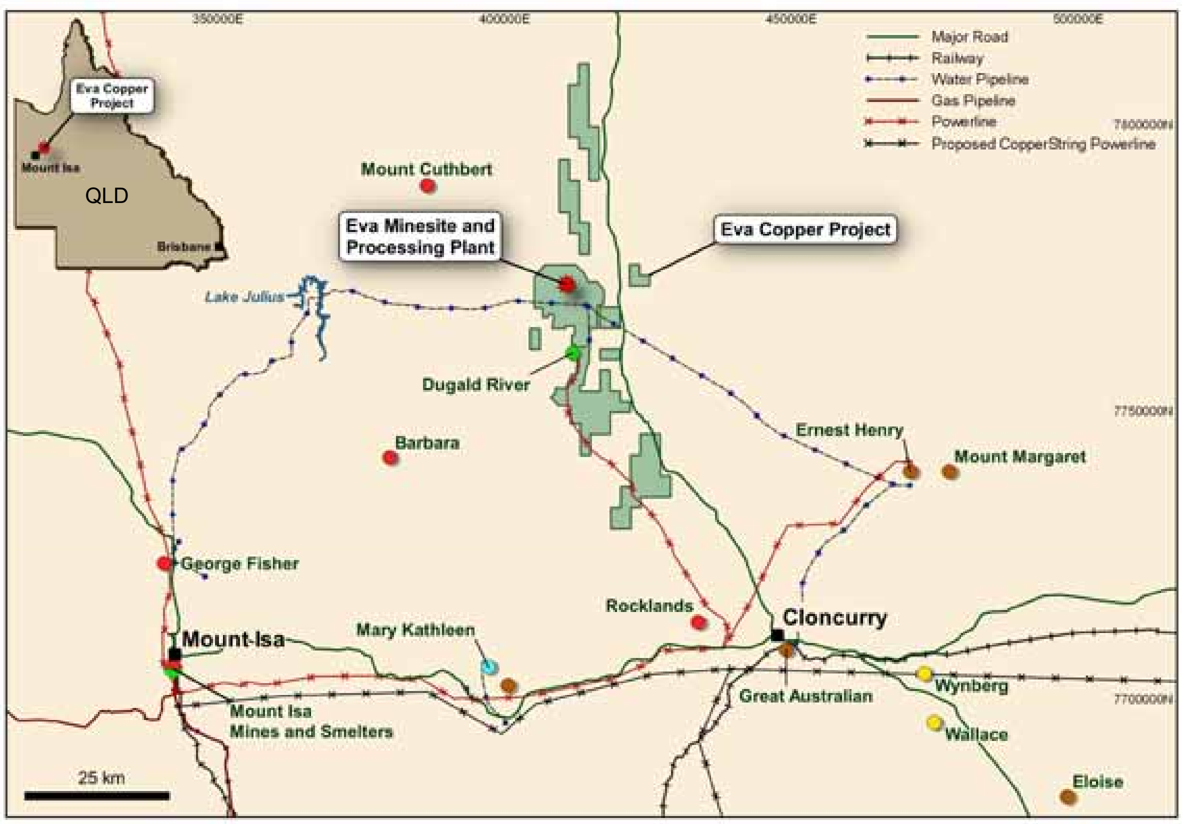


Figure 2-1: Location, Tenure, Plant, and Regional Infrastructure

The Project is proposed to be a large, open pit copper-gold mining operation with an associated gravity and flotation processing plant, similar to other operations in the Mount Isa and Cloncurry area. The Project comprises the large Little Eva open pit and six smaller satellite pits, which will deliver an ore mixture with a maximum of 25% native copper ore to a 11.388 million tonnes per annum (Mt/a) processing plant adjacent to the Little Eva and Turkey Creek pits.

The Little Eva deposit was the subject of a major drilling program from 2010 to 2012, which consequently more than doubled the deposit's contained Mineral Resources. The enlarged Little Eva deposit was the focus of many feasibility studies, including the CMMC Feasibility Study in 2018 comprising a simple operation treating copper-gold sulphide ore. However, in 2019, CMMC performed additional infill drilling to the Blackard deposit confirming substantial historical exploration and metallurgical work, and subsequently included the Blackard and Scanlan deposits in this updated Technical Report, improving the Project reserves by 45% (from 117 Mt to 170 Mt) the Project net present value (NPV) by 66% (from $256 million to $425 million) and the life-of-mine (LOM) recoverable copper by 57% (from 959 million pounds (Mlb) to 1,505 Mlb).

The process plant redesign was guided by recent additional metallurgical testwork, and is in several ways similar to the Company's existing Copper Mountain Mine processing plant near Princeton, British Columbia, Canada. It is also similar to the New Afton mine near Kamloops in British Columbia, Canada, and the Ernest Henry mine in Queensland, which is 60 km distance from the Eva Copper Project.

It is estimated that over 28 years, a total of $46.9 million has been expended on exploration, resource development, metallurgical and engineering studies, compensation payments, government fees, and charges by Altona's predecessor, Universal Resources Limited (Universal), Universal's partners, and by parties who held the Project prior to Universal. Altona spent approximately $21.0 million from February 2010 through March 2018, and CMMC has spent $4.8 million since taking ownership of CMMPL.

Responsible for specific report sections, the qualified persons (QPs) as defined under NI 43-101 (by virtue of their education, experience, and professional association, and their membership or good standing with appropriate professional institutions or associations) are as follows:

- Paul Staples, Vice President and Global Practice Lead, Ausenco Engineering Canada Inc. (Ausenco)
- Alistair Kent, Senior Project Manager, Merit Consultants International (Merit)
- David Johns, Senior Geotechnical Engineer, Klohn Crippen Berger (KCB)
- Peter Holbek, Vice President Exploration, Copper Mountain Mining Corp. (CMMC)
- Stuart Collins, Mining Consultant, SEC Enterprises Corp. (SECEC)
- Mike Westendorf, Director Metallurgy, Copper Mountain Mining Corp. (CMMC)
- Roland Bartsch, Vice President and Country Manager Australia, Copper Mountain Mining Pty. Ltd. (CMMPL)
- Richard Klue, Vice President Technical Services, Copper Mountain Mining Corp. (CMMC).

2.1 Copper Mountain Mining Corporation

The Company has a flagship asset, the Copper Mountain Mine located in southern British Columbia, Canada, near the town of Princeton. The Company has a strategic alliance with Mitsubishi Materials Corporation, who owns 25% of the Copper Mountain Mine. The Copper Mountain Mine is large copper-gold porphyry that produces on average approximately 80 Mlb of copper annually over its 31-year mine life. The Copper Mountain Mine has a large copper resource that remains open laterally and at depth, including the New Ingerbelle deposit, which it continues to explore to fully appreciate the property's development potential.

2.2 Terms of Reference and Purpose

This report is prepared as an NI 43-101 Technical Report for CMMC by Ausenco, Merit, and KCB, and will be filed with the Canadian Securities Administrators (CSA) in the System for Electronic Document Analysis and Retrieval (SEDAR) filing system.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort based on:

- Information available at the time of preparation
- Data supplied by outside sources
- The assumptions, conditions, and qualifications set forth in this report.

2.3 Report Section Responsibilities

Table 2-1 shows a list of all the sections included in this NI 43-101 Technical Report, and the respective QPs from Ausenco, Merit, and KCB that assisted CMMC's QPs in compiling this report.

Table 2-1: Scope of Responsibility

Item	Content	Qualified Person	Compiled by
1	Summary	PS, AK, DJ, PH, SC, MW, RB, RK	All
2	Introduction	RK	CMMC
3	Reliance on Other Experts	PS, AK, DJ, PH, SC, MW, RB, RK	All
4	Property Description and Location	RB	CMMPL
5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	RB	CMMPL
6	History	RB	CMMPL
7	Geological Setting and Mineralization	PH	CMMC
8	Deposit Types	PH	CMMC
9	Exploration	PH	CMMC
10	Drilling	PH	CMMC
11	Sample Preparation, Analyses, and Security	PH	CMMC
12	Data Verification	PH	CMMC
13	Mineral Processing and Metallurgical Testing	MW	CMMC
14	Mineral Resource Estimates	PH	CMMC
15	Mineral Reserve Estimates	SC	SECEC
16	Mining Methods	SC	SECEC
17	Recovery Methods	PS	Ausenco

Item	Content	Qualified Person	Compiled by
18	Project Infrastructure	RK, DJ, PS	CMMC, KCB, Ausenco
19	Market Studies and Contracts	RK	CMMC
20	Environmental Studies, Permitting, and Social or Community Impact	RB	CMMPL, MBS
21	Capital and Operating Costs	AK, SC, MW	Merit, SECEC, CMMC
22	Economic Analysis	RK	CMMC
23	Adjacent Properties	RB	CMMPL
24	Other Relevant Data and Information	AK	Merit
25	Interpretation and Conclusions	PS, AK, DJ, PH, SC, MW, RB, RK	All
26	Recommendations	PS, AK, DJ, PH, SC, MW, RB, RK	All
27	References	PS, AK, DJ, PH, SC, MW, RB, RK	All
28	Certificates of Qualified Persons	PS, AK, DJ, PH, SC, MW, RB, RK	All

Notes: Qualified Persons and their acronyms are listed below. The following individuals, by education, experience, and professional association, are considered QPs as defined in the NI 43-101 Standards of Disclosure for Mineral Projects, and they are members in good standing with appropriate professional institutions or associations. The QPs are solely responsible for the specific report sections listed with their abbreviations in Table 2-1.
PS – Paul Staples, Vice President and Global Practice Lead, Ausenco
AK – Alistair Kent, Senior Project Manager, Merit
DJ – David Johns, Senior Geotechnical Engineer, KCB
PH – Peter Holbek, Vice President Exploration, CMMC
SC – Stuart Collins, Independent Mining Consultant, SECEC
MW – Mike Westendorf, Director Metallurgy, CMMC
RB – Roland Bartsch, Vice President and Country Manager, Australia, CMMPL
RK – Richard Klue, Vice President Technical Services, CMMC.

2.4 Sources of Information and Data

This Technical Report, as input for its analysis, site and process plant design, and material take-offs (MTOs), relies largely on inputs from Ausenco, CMMC, CMMPL, Merit, SECEC, KCB, Knight Piésold, MBS Environmental (MBS), and Rockwater Hydrogeological Consultants (Rockwater), Paterson & Cooke (P&C), CITIC SMCC Process Technology, Pty. Ltd. (CITIC SMCC), KH Morgan and Associates, (Morgan), Metso Corporation (Metso), Gekko Systems (Gekko) and Woodgrove Technologies (Woodgrove), These sources of data referred to in Section 27.

2.5 Personal Inspection

In accordance with NI 43-101 guidelines, the Resource Estimate and Reserve Estimate QPs (Peter Holbek and Stuart Collins) individually visited the Project site in February 2015 and September 2018, respectively, in the Company of CMMPL staff in Australia. Mr. Holbek's site visit was led by Mr. Roland Bartsch. Mr. Bartsch has been associated with the development team of the Project for five years. Mr. Collins visited the Project from September 21 to 24, 2018. Mr. George Ross, Chief Geologist of CMMPL, led Mr. Collins' site visit.

During the site visits, the QPs inspected outcrops of the Little Eva, Bedford, Turkey Creek, Lady Clayre, Ivy Ann, Blackard, and Scanlan deposits. The QPs also viewed core samples of representative diamond drill holes (DDH) over a selection of orebodies. Several drill hole collars were verified with a Global Positioning System (GPS) instrument. Independent quality control (QC) samples were not taken, but the visual nature of the copper mineralization was apparent both in outcrop and in core. Manual checking of the assays in the database against the original assay certificates was carried out by the QPs at that time. CMMC QP Mike Westendorf, who is responsible for Section 13, visited the site in July 2019, touring the Project site and adjacent operating properties. Paul Staples, who is responsible for Section 17, did not visit the site during the course of preparing and compiling the report sections, and relied solely on data published in previous reports issued by CMMC, CMMPL, Gekko, Paterson & Cooke, Metso, ALS Metallurgy (formerly AMMTEC), Optimet, NeoProTec Pty. Ltd., Larox, and GR Engineering Services.

Additional visits to the Eva Copper Project site were made by Gil Clausen (CEO, CMMC) and Don Strickland (COO, CMMC) in July 2018. Richard Klue (VP Technical Services, CMMC) and Lance Newman (VP Project Development, CMMC), who each have extensive mining, projects, and metallurgical expertise, conducted a site visit in October 2018.

2.6 Effective Date

The effective date of the Mineral Resources and Mineral Reserves statement in this report is January 31, 2020. There have been no material changes to the Mineral Resources and Mineral Reserves since that date.

The NI 43-101 Technical Report date is May 7, 2020.

2.7 Abbreviations and Units of Measure

Units of measure used in this report conform to the metric system, unless noted otherwise. All currency is United States dollars (US$) unless noted otherwise. A glossary containing a comprehensive list of acronyms and units of measure is included in Section 27.

3 RELIANCE ON OTHER EXPERTS

The QPs' opinions contained herein are based on public and private information provided by Copper Mountain Mining Corp. (CMMC) and others throughout the course of the study. The authors have carried out due diligence reviews of the information provided to them by CMMC and others for preparation of this report. The authors are satisfied that the information was accurate at the time of writing and that the interpretations and opinions expressed are reasonable and are based on a current understanding of the mining and processing techniques and costs, economics, mineralization processes, and the host geologic setting. The authors have made reasonable efforts to verify the accuracy of the data relied on for this report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Eva Copper Project is located 76 km by road northwest of Cloncurry, a town of about 3,000 inhabitants, and 194 km by road from Mount Isa, a regional mining centre with a population of about 22,000 people (Figure 4-1). Townsville on the east coast is 770 km from Cloncurry. Access to the Project is from the sealed Burke Developmental Road, which originates in Cloncurry. This road passes 8.5 km to the east of the proposed plant site, and current access is via cattle station and exploration tracks. The planned site for the plant and major infrastructure is also 11 km north of the major Dugald River Zinc Mine, which was commissioned in November 2017 and is owned by MMG Limited (MMG).

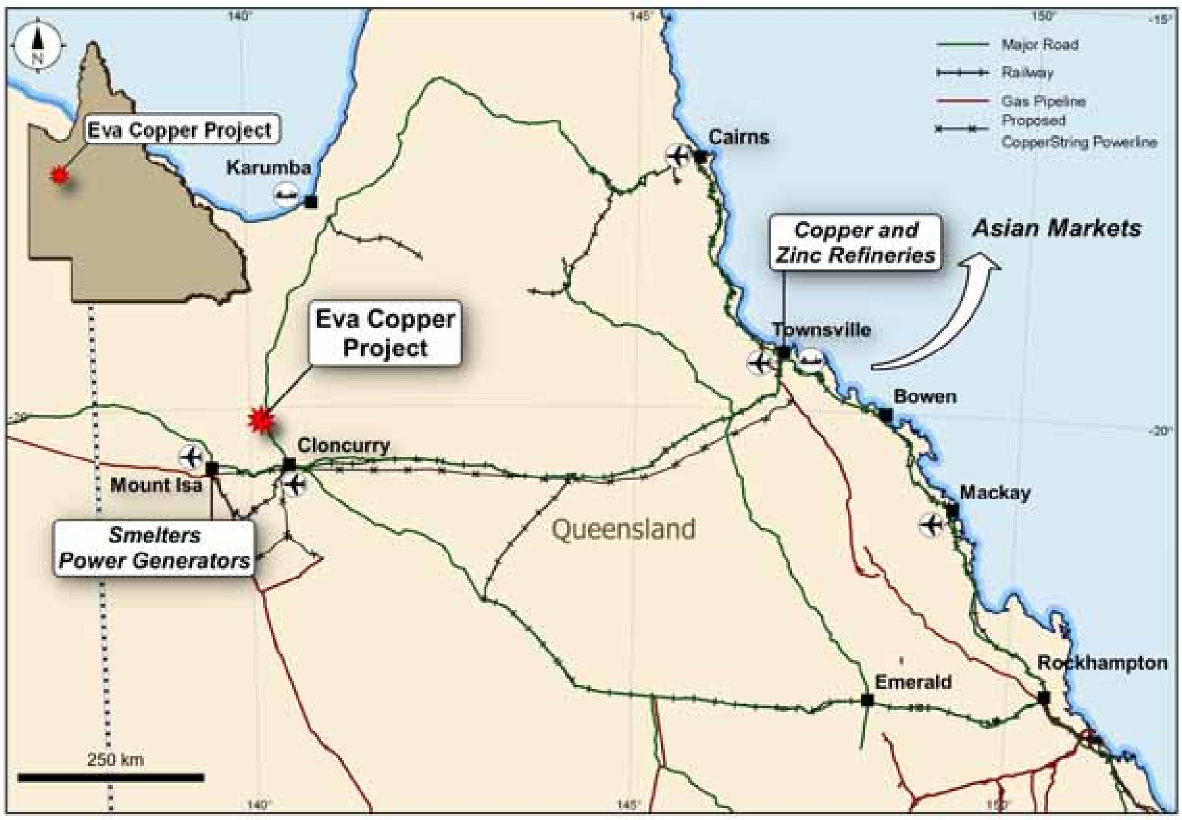


Figure 4-1: Project Location

4.2 Land Use and Mining Tenure

The Eva Copper Project consists of five Mining Leases (ML) and one Exploration Permit for Minerals (EPM). All six of the planned pits are located within the MLs, except for the Ivy Ann pit, which lies within EPM 25760 (King).

Queensland state legislation requires that, where significant disturbance will occur from exploration and mining activities, the license holder must reach agreement for "Conduct and Compensation" with the pastoral leaseholder. CMMPL, has secured such agreements for all the MLs, the Ivy Ann deposit, and those portions of the EPM where ground disturbance has occurred or is anticipated.

4.3 Mining Leases

The MLs were granted in 2012 and are currently owned by the Company's wholly owned subsidiary Eva Copper Mine Pty. Ltd. (ECMPL). The MLs total area is 143 km^2 and are situated across from two pastoral lease holdings and within one Native Title determination area.

Table 4-1: Eva Copper Project Mining Leases

Number	Name	Granted	Expiry	Area (ha)
90162	Scanlan	Oct. 4, 2012	Oct. 31, 2037	2,096.96
90163	Longamundi	Oct. 4, 2012	Oct. 31, 2037	1,411.29
90164	Blackard	Nov. 13, 2012	Nov. 30, 2037	5,131.07
90165	Little Eva	Nov. 13, 2012	Nov. 30, 2037	5,029.96
90166	Village	Nov. 13, 2012	Nov. 30, 2037	616.08

4.4 Exploration Permits for Minerals

As shown in Table 4-2, the Company's wholly owned subsidiary ECMPL holds the EPM 25760 (King), which encompasses the Ivy Ann deposit.

Table 4-2: Eva Copper Project Exploration Permit for Minerals

Number	Name	Holder	Granted	Expiry	Area (ha)
25760	King	ECMPL	Nov. 17, 2015	Nov. 17, 2020	28,601

The Company also holds 26 EPMs surrounding the MLs and in the broader Mount Isa region (Figure 4-2). These are held by the Company's wholly owned subsidiaries Roseby Copper Pty. Ltd. and Roseby Copper (South) Pty. Ltd. (RCSPL).

Agreements exist with four pastoral landholders for both the MLs and key areas of activity in the surrounding EPMs:

- Coolullah Station, belonging to the North Australian Pastoral Company (NAPCO)
- Mt. Roseby Station, belonging to Harold Henry McMillan
- Dipvale Station, belonging to Grant and Anita Telford
- Hillside Station, belonging to the Cameron Creek Pastoral Company.

The locations of the Pastoral Lease boundaries intercepted by the Project tenements and various mineralized areas are shown in Figure 4-3; in relation to the Project tenements and the areas subject to Conduct and Compensation Agreements.

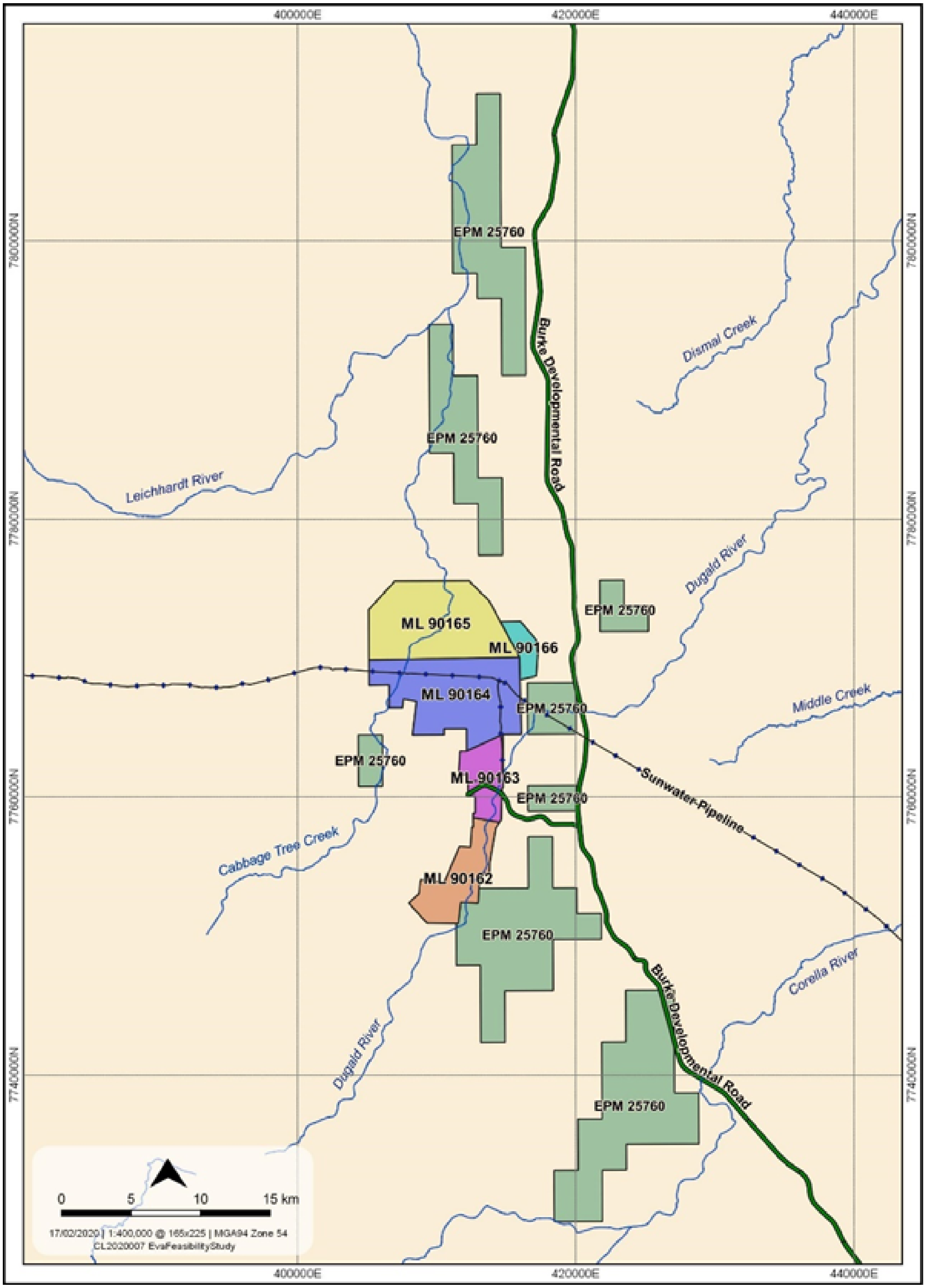
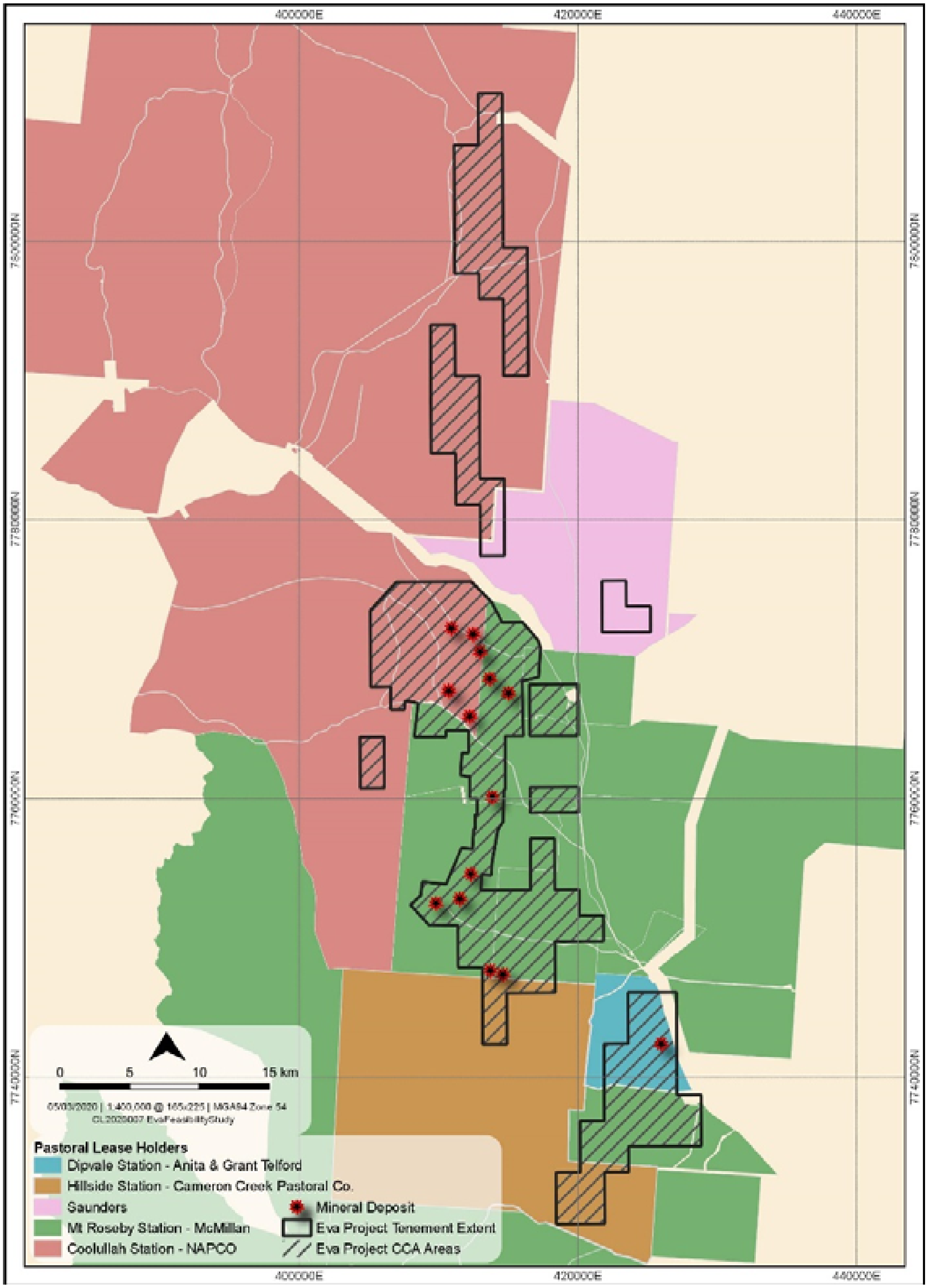


Figure 4-2: Eva Copper Project Tenements



Notes: (NAPCO (red-brown), McMillan (green), Telford (blue) and Cameron Creek Pastoral Company (brown) showing deposits (mine symbols))

Figure 4-3: Pastoral Lease Holdings and Current Conduct and Compensation Agreement Areas (Colour indicates landowner)

4.4.1 Coolullah Station (NAPCO)

The Little Eva, Turkey Creek, and Blackard (in part) pits, the proposed mill, a large portion of the tailings storage facility (TSF), and other infrastructure lie within Coolullah Station. Two sets of compensation agreements have been agreed upon, one for the portion of the MLs that fall within the station, the second for the Eva Project EPM and surrounding EPMs.

The key commercial terms of the agreements are:

- Mining Leases (Agreement Numbers 349 and 1078):
  - Term of Agreement: from July 21, 2008, to expiry of MLs.
  - Compensation: $20,000 on issue of MLs, $119,060 on commencement of any activities. This compensation applies to a predetermined area (First Area) of approximately 740 ha set out in the agreement and selected based on the 2005/2009 DFS mine layout footprint, which differs significantly from the current layout. Additional areas outside this area will incur compensation of $100/ha, indexed. Replacement of water sources ($70,000).
  - MLs 90164 and 90165 lie partially within Coolullah Station.
    The "commencement of activities" trigger for compensation occurred, and the compensation payment was paid on August 1, 2014. This payment was based on an area specified by an earlier mine layout and location (DFS).
- Exploration Permits (Agreement Numbers 1110, 1111 and 112):
  - Term of Agreement: 24 months, expiry 12 September 2021.
  - Compensation: annual payments and payments for disturbance of $500/ha total disturbed area, indexed annually.
  - EPM 25760 lies partially within Coolullah Station and is covered by Agreement Number 112.

4.4.2 Mt. Roseby Station (McMillan)

The planned Bedford, Lady Clayre, Blackard (in part), and Scanlan pits, and part of the TSF lie within Mt. Roseby Station. Compensation Agreements for both the MLs and the EPM have been entered into with the owner, Mr. Harold McMillan. The key commercial terms of the agreements are:

- Mining Leases (Agreement Numbers 396 and 1079):
  - Term of Agreement: from June 30, 2008 to expiration of MLs.
  - Compensation: $20,000 on signing agreement; $500,000 within 30 days of Work Area Occupation date; $600,000 on the first anniversary of $500,000 payment date, $550,000 on the second anniversary, and $500,000 on the third anniversary; and $600,000 within 30 days of Scanlan occupation date. Additional areas greater than 1,500 ha are to be compensated at a rate of $250/ha, indexed. The Work Area corresponds to plant and camp areas, with locations specified in the agreement and based upon the 2005/2009 DFS. The current feasibility study mine layout has changed, with the plant relocated onto Coolullah Station, a large area of the TSF extended onto Mt. Roseby Station, and the ML access road relocated but still on Mt. Roseby Station. Agreement terms include stock-proof fencing of roads and mine areas, cattle crossings, and stock water sites. Due to the proximity to Mt. Roseby Homestead, activities in the Scanlan area trigger provisions that include longer notice periods and soundproofing works (modification of residences and a pit bund).
  - MLs 90162, 90163, and 90166 lie wholly within Mt. Roseby Station. MLs 90164 and 90165 lie partially within Mt. Roseby Station.

- Exploration Permits (Agreement Number 992 and 1079)
  - Term of Agreement: from July 1, 2017 to July 1, 2022.
  - Compensation: Annual Compensation of $2,000 per tenement per year, and a drilling work rate of $250 per reverse circulation (RC) or diamond drill hole (DDH) completed within a financial year. The annual compensation and drilling work rate will be adjusted annually in accordance with the consumer price index (CPI) on July 1 each year during the Term.
  - EPM 25760 lies partially within Mt. Roseby Station.

4.4.3 Dipvale Station (Telford)

The Ivy Ann deposit lies within Dipvale Station, and a Conduct and Compensation Agreement relating only to exploration activity has been signed. An additional compensation agreement would be required to support any future ML application; however, an ML is not required at this stage. The key commercial terms are:

- Exploration Permit (Agreement Numbers 623 and 1071)
  - Term of Agreement: from August 30, 2011 for the term of the tenements, including renewals.
  - Compensation: $10,000 on signing agreement, plus $1,000/ha of land disturbed in each financial year. Compensation increases at 4% each year.
  - EPM 25760 lies partially within Dipvale Station (replacing EPM 8059, as set out in the agreement).

4.4.4 Hillside Station (Cameron Creek Pastoral Company)

Only exploration tenure falls within Hillside Station. A Conduct and Compensation Agreement has been signed. They key commercial terms are:

- Exploration Permit (Agreement Number 1063)
  - Term of Agreement: from May 28, 2018 to May 27, 2023 (five years).
  - Compensation: $5,500 annually, plus $750 for each hectare of land disturbed (CPI adjusted).
  - Payments relate to exploration activity only.
  - Tenements: EPM 25760 lies partially within Hillside Station. Other tenements within the agreement are held by RCSPL and are not part of the Project.

4.5 Freehold Land

Two freehold lots that were granted in the late 1800s sit within the MLs. One sits over part of the Little Eva deposit, the second over part of the Longamundi deposit.

4.5.1 Lot 37 (Agreement Numbers 355, 526, 1069, and 1070)

Lot 37 (on Crown Plan B15752) is located within ML 90165 and overlies the Little Eva deposit (Figure 4-4). It is owned 100% by the Company; 50% was purchased from Pasminco (referred to as Mineral Selection 3072), and 50% deeded to the Company by The Public Trustee of Queensland from an intestate deceased estate. The Lot was previously subject to mining tenure Mineral Development Licence 12 (also purchased from Pasminco), and has also been referred to as the Kwahu Moiety area.

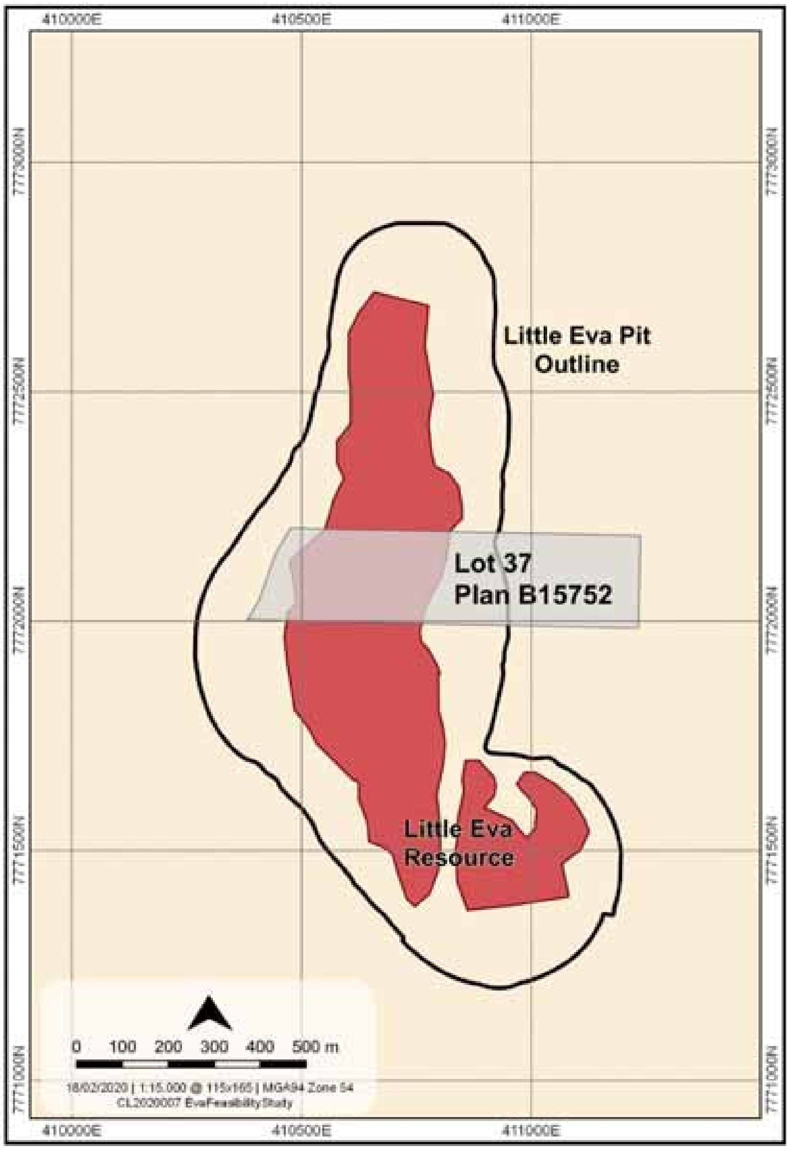


Figure 4-4: Location of Lot 37 over the Little Eva Deposit

4.5.2 Lot 28 (Agreement Numbers 355, 1069, and 1070)

Lot 28 (on Crown Plan B15753) is located within ML90163, overlies the Longamundi deposit, and is owned 100% by the Company. It was purchased from Pasminco (referred to by them as Mineral Freehold 13961). The lot was previously subject to ML 7497 (also purchased from Pasminco).

4.6 Other Third-Party Access and Compensation Agreements

In addition to reaching agreement on compensation and access with pastoral leaseholders, various third-party agreements and consents are required by the Queensland Government to secure the grant of MLs and use of ML Access Road. The relevant agreements are detailed below.

4.6.1 Mining Compensation Agreement – Minister for Transport and Multicultural Affairs (Agreement Numbers 625 and 1103)

Access to the ML area requires crossing a rail corridor adjacent to the main public road. In 1995, this land was leased to the state in perpetuity. Section 279 of the Mineral Resources Act requires compensation to be paid to the State for provision of access. The agreement refers to access to cross a decommissioned narrow-gauge rail line (Lot 251 at the northern boundary of Lot 6 on GR33) that runs from Cloncurry to Kajabbi, which lies to the east of the MLs and is adjacent to the sealed Burke Developmental Road.

4.6.2 Access Licence – Department of Main Roads (Agreement Numbers 626 and 1104)

The Department of Main Roads is the registered lessee of part of Lot 211 on SCP 136468, which forms part of the Cloncurry–Kajabbi rail corridor. The license provides for use and operation of a roadway over this land to access the MLs.

4.6.3 Compensation and Consent Agreement – Minister for Natural Resources (Agreement Numbers 482 and 1098)

The State owns Lot 6 on CP GR33 in the Parish of Merkara, County of Granada, in the State of Queensland, which is a Reserve for the purposes of the *Mineral Resources Act*. The agreement allows for use of the land in accordance with section 279 of the Minerals Act. The land covers the eastern section of the mine access road where it joins the sealed Burke Developmental Road, surrounding the historical rail station site on Lot 211.

4.6.4 MMG Access and Other Agreements (Agreement Numbers 367, 368, 369, 422, 423, 604, and 1075)

The Company has several agreements with MMG, which, among other things, provide for MMG to apply for the various licenses required to locate infrastructure for the Dugald River zinc mine within the Company's mining tenure. Under the agreement, MMG has constructed a sealed access road, water pipeline, and a power line within the Company's tenure.

4.6.5 SunWater Limited – Indemnity (Agreement Numbers 668 and 1083)

SunWater Limited (SunWater) is the registered grantee of the water supply easement along which the Lake Julius to Ernest Henry pipeline is located. The agreement provides the Company limited access across the pipeline and easement to access the MLs and conduct mining activities.

4.6.6 Cloncurry Shire Council (Agreement Numbers 429, 1080, and L10523)

Although the Cloncurry Shire Council does not maintain any road infrastructure within the Project area, the Council is the owner, for the purposes of the *Mineral Resources Act*, of a number of roads and other areas within the MLs. This agreement provides for the grant of access over the deemed Council property to access the MLs (including Lot 6 on GR33 for the ML Access Road).

4.7 Royalties

Numerous royalties apply to the Project area, and are payable to six parties, as described below. Each royalty has only been described to the extent that it pertains to the Project area. There may be other royalty obligations, with respect to other tenures held by the Company, contained within the same agreements.

4.7.1 State of Queensland

Royalties on minerals are payable annually to the Queensland State Government through the Department of Natural Resources and Mines on an ad valorem basis, with various costs being permitted as a deduction from sales revenue.

Copper and gold royalty rates vary between 2.5% and 5.0% of value, depending on average metal prices, as per Schedule 3 of the *Mineral Resources Regulation* of 2013. No royalty is payable on the first $100,000 of the combined value of certain minerals sold, disposed of, or used in a financial year (the royalty-free threshold); the threshold applies to both copper and gold. A royalty discount applies for base metals processed within Queensland to a particular metal content. Copper processed to 95% metal content through leaching, smelting, and solvent extraction electrowinning receives a 20% discount applied to the royalty payable. The royalty discount is applicable to the Project where the copper concentrate is being processed at the Mt. Isa smelter under the terms of an offtake agreement with Glencore International AG.

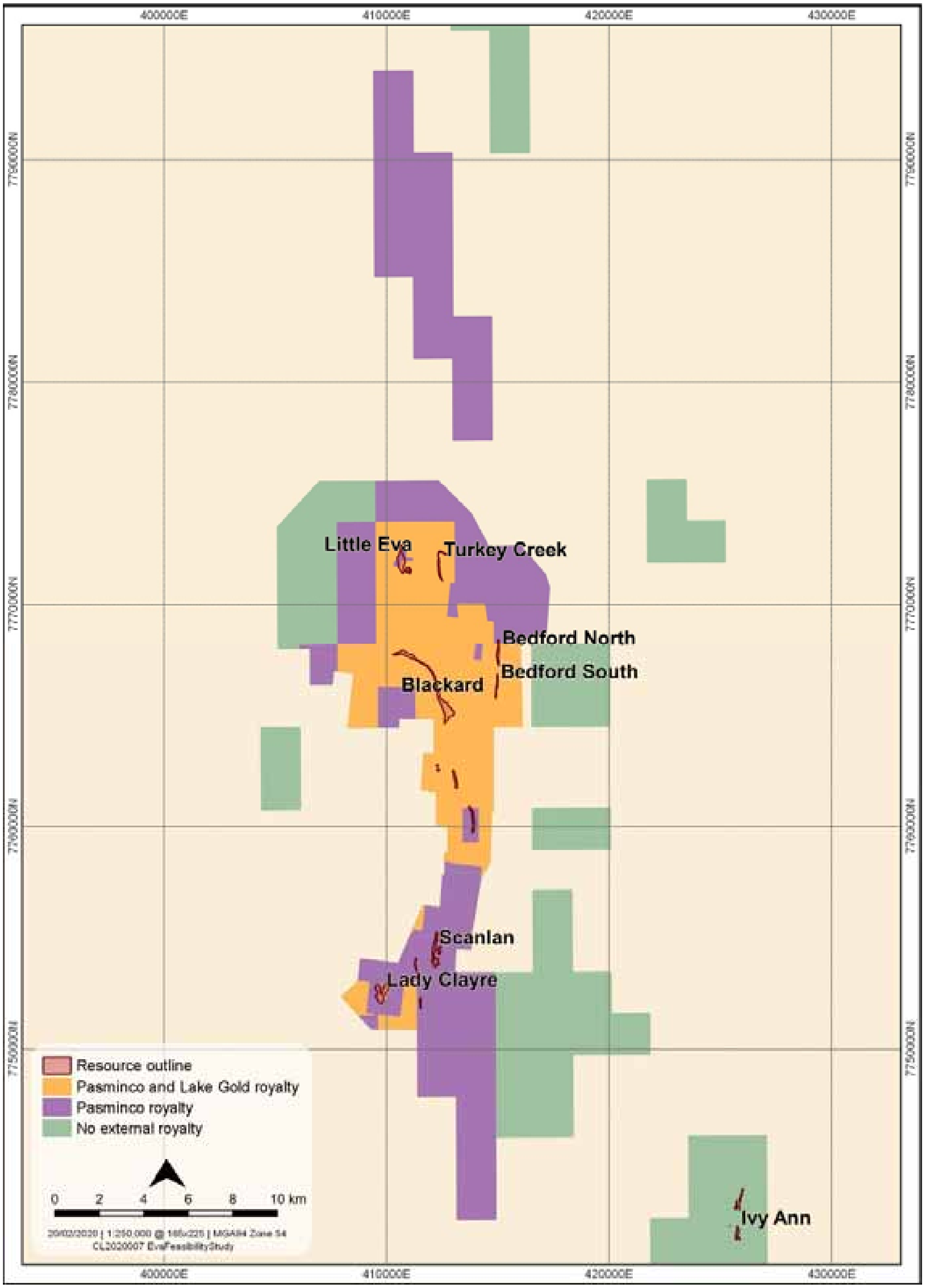
Where freehold land was held prior to 1904, all Mineral Resources were owned by the titleholder. The Company owns two freehold titles (Lot 28 (Longamundi), and Lot 37 (Little Eva pit)) (see Section 4.4) for which no state royalty is applicable. Gold may have been treated differently than other minerals; it has not been confirmed if a royalty on gold is payable to the state on these freehold titles.

4.7.2 Pasminco Royalty (Agreement Numbers 355, 1076, and 1077)

A portion of EPM 25760 and a significant area of the MLs that were purchased from Pasminco are subject to a royalty, now payable to MMG. The area subject to this royalty corresponds to superseded MLs and EPMs set out in Agreement Number 355. A 1.5% net smelter return (NSR) royalty applies to this area.

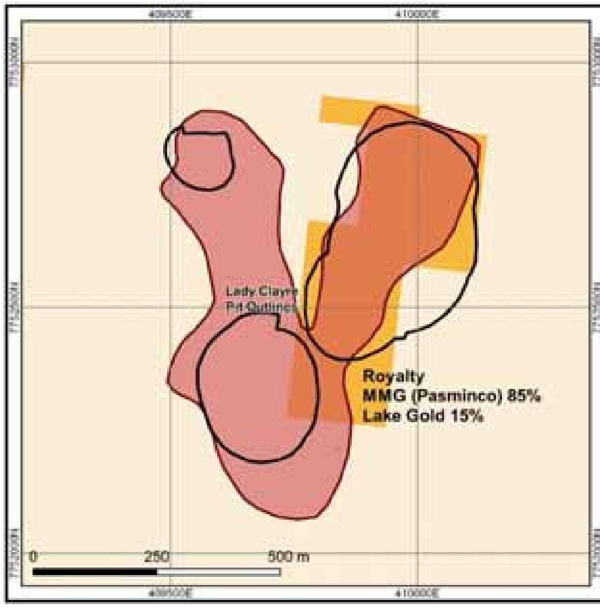
The area of the Project subject to this royalty is illustrated in Figure 4-5, and it includes parts of the Little Eva, Bedford, and Lady Clayre deposits, and all of the Scanlan deposit. Various first rights of refusal also apply.

Lot 37 was included in the acquisition from Pasminco. A royalty stream referred to as the Kwahu Moiety sits over Lot 37 (see Section 4.4), and is evidenced by an agreement between CRA Exploration (CRAE) and The Kwahu Company Limited (Agreement Number 476). The Kwahu Moiety obligations were not assigned to the Company when they acquired their interests from Pasminco and Lake Gold.



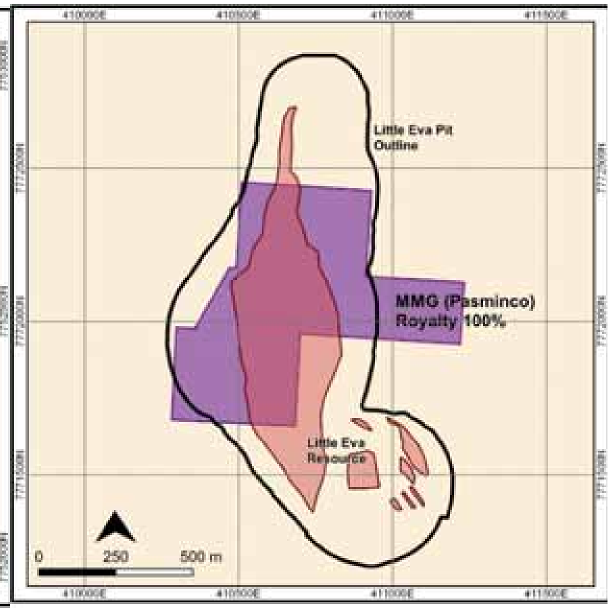
Note: For detail of the Lady Clayre and Little Eva deposits, refer to Figure 4-6 and Figure 4-7

Figure 4-5: 100% MMG (Pasminco) and 85% MMG (Pasminco) / 15% Lake Gold Agreement Areas



Note: Mineral Reserves outside the area marked in orange are 100% MMG (Pasminco)

Figure 4-6: 85% MMG (Pasminco) / 15% Lake Gold Royalty Area Over Lady Clayre Deposit



Note: Mineral Reserves outside the area marked in purple are 85% MMG (Pasminco)/15% Lake Gold

Figure 4-7: 100% MMG (Pasminco) Royalty Area Over the Little Eva Deposit

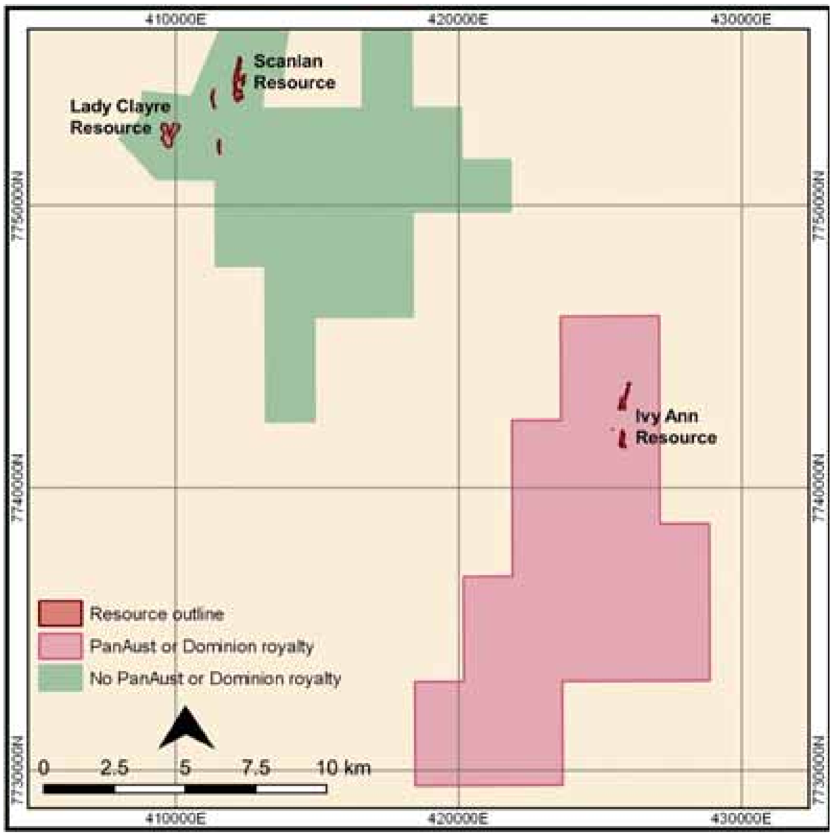
4.7.3 Pasminco-Lake Gold Royalty (Agreement Numbers 355, 463, 485, 1076, and 1077)

A portion of the area purchased by Universal from Pasminco is subject to a 1.5% NSR royalty payable to Lake Gold (15%) and Pasminco (now MMG, 85%).

The area of the Project subject to this royalty is shown in Figure 4-5 and includes parts of the Little Eva, Bedford, and Lady Clayre deposits, and all of the Blackard and Turkey Creek deposits.

4.7.4 PanAust (Agreement Numbers 438, 439, 440, 509, 512, 511, 510, 968, and 1072)

A portion of EPM 25760 (corresponding to the area of superseded EPM 8059) purchased by Universal from Pan Australian Resources NL (PanAust) and Dominion is subject to a 1.6% NSR royalty payable to PanAust. EPM 8059 was consolidated with other EPMs through conditional surrender in favour of EPM 25760. The Ivy Ann deposit lies within what was EPM 8059 (now part of EPM 25760), and is subject to a 1.6% NSR royalty to PanAust.



Note: Corresponding to superseded EPM 8059

Figure 4-8: Area Subject to the Dominion and PanAust Royalties

4.7.5 Dominion (Agreement Numbers 388, 426, 427, 428, 484, 562, and 1074)

A portion of EPM 25760 (corresponding to superseded EPM 8059) purchased by Universal from PanAust and Dominion is subject to a 0.4% NSR royalty payable to Dominion in addition to the PanAust 1.6% NSR royalty listed in Section 4.7.4.

EPM 8059 was consolidated with other EPMs through conditional surrender in favour of EPM 25760. The Ivy Ann deposit lies within what was EPM 8059 (now part of EPM 25760), and is subject to an NSR royalty of 0.4%.

4.7.6 Kalkadoon People (Agreement Numbers 415, 416, and 1099)

A 0.22% NSR royalty is provided as compensation for the effects of mining activities on the Native Title of the Kalkadoon People.

4.7.7 Mt. Isa Mines (Agreement Numbers 661 and 1073)

A royalty of AUS$0.50 per dry tonne (dmt) of product mined from the tenement area and processed on or outside the tenement area is payable to Mt. Isa Mines on any production from a portion of EPM 25760 (King) corresponding to superseded EPM 13249 (Lilliput, Figure 4-9). These are exploration tenements only, and no Mineral Resources or Mineral Reserves have yet been defined or included in this Feasibility Study; therefore, no royalty payment will be required at this time.

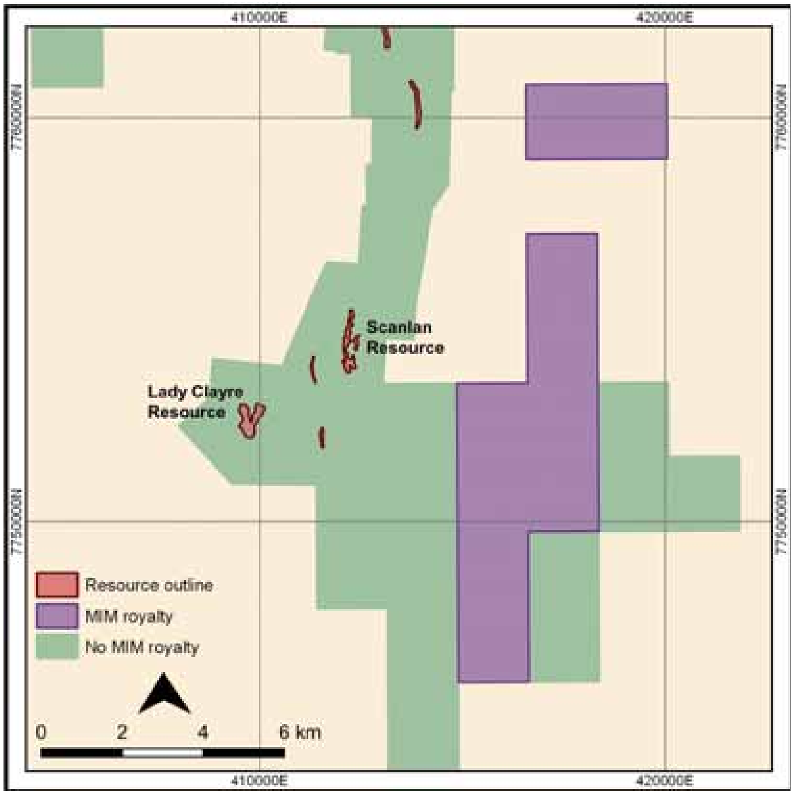


Figure 4-9: Project Area Subject to Mt. Isa Mines Royalty

4.7.8 Royalty Summary

Table 4-3 summarizes the royalties applicable to various deposits.

Table 4-3: Royalties Applicable to Portions of the Mineral Reserves at Various Deposits

Deposit	Area	State	MMG	Lake Gold/MMG	KD	PanAust	DOM
Little Eva	Lake Gold	x		x	x		
Little Eva	Freehold		x		x		
Little Eva		x	x		x		
Blackard		x		x	x		
Scanlan		x	x		x		
Turkey Creek		x		x	x		
Lady Clayre	Lake Gold	x		x	x		
Lady Clayre		x	x		x		
Bedford		x		x	x		
Ivy Ann		x			x	x	x

Notes: KD = Kalkadoon; DOM = Dominion

4.8 Environmental and Permitting

To the extent known, the key environmental considerations and permits required for a mining project in Queensland (described in more detail in Chapter 20) are:

- Tenure (ML) from the Department of Natural Resources, Mines and Energy (DNRME) that gives access to the land
- An Environmental Authority (EA) from the Department of Environment and Science (DES) that regulates the environmental management of the Project.

Both the MLs and the EA have been approved. The current EA is based on the previous 2016 mine layout; changes to the mine layout will require submission of Amendments. Amendments are assessed to determine if they are "Minor" or "Major." From the date of application submission, the Minor Amendment process takes up to 35 days, while the time for a Major amendment can vary. The 2016 Major Amendment for Altona took 3.5 months from the date of application submission. Changes to the mine layout, extent of the new footprint and increased ore processing throughputs in this feasibility study will require a Major Amendment.

The EA sets out key environmental management conditions and should be referred to for full details.

The Queensland Government introduced rehabilitation and Financial Assurance (FA) reforms subsequent to grant of the current EA and previous feasibility study that included the *Mineral and Energy Resources (Financial Provisioning) Act 2018* (MERFP Act) that was passed in November 2018. New regulatory requirements result from the reforms and are included here.

Key EA regulatory management issues, particularly in the mine development period, are:

- EA Major Amendment application. The current EA is based on a previous 2016 mine layout. Changes to the mine layout will require submission of an EA Major Amendment to the DES. This is a straightforward requirement that with application preparation and pre-lodgement meetings.
- Progressive Rehabilitation and Closure (PRC) plan submission. Organizations carrying out mining activities in Queensland are legally obligated to rehabilitate the land. Recent legislation reforms require holders of an existing EA for a mining activity relating to a mining lease approved through a site-specific application granted prior to passage of the PRC plan legislation (as per Eva), to develop and submit a PRC plan to the DES. As mine development at Eva has not commenced, a PRC plan is required to be submitted in conjunction with the proposed EA Major Amendment application.
- Estimated Rehabilitation Cost (ERC) decision. An ERC decision is required to be in effect before commencing any activities under an EA. The ERC is the estimated cost of rehabilitating the land on which a resource activity is carried out, and preventing or minimizing environmental harm, or rehabilitating or restoring the environment in relation to the resource activity. DES is responsible for deciding the ERC for an EA for resource activities. The ERC came into effect in 2019 under the MERFP Act reforms, and replaces the previous Plan of Operations (PoO) requirements.
- ERC scheme Financial Assurance (FA). This is required to be lodged with DES (either as a contribution paid to the scheme fund, or as a surety given under the MERFP Act) prior to any activities being allowed to commence. The amount of the FA required is calculated in accordance with DES procedures, based on the implementation of site-specific rehabilitation and closure tasks, using independent contractor third-party rates. The amount of the FA is directly related to the activities authorized.

- Design plan for the Cabbage Tree Creek diversion. Final detailed plans will need to be formally submitted, and approval received, prior to construction being allowed to commence.
- Environmental offset requirements. The Project triggers the requirement of an offset due to the disturbance of regional ecosystems resulting from the disturbance of Cabbage Tree Creek. There are two options for offsets: a financial settlement, or a proponent-driven offset which may include approved conservation work programs. A series of submissions are required, including Significant Impact Details, Offset Report, and Notice of Election at least four months prior to commencement of any site work (Significant Residual Impacts). To fulfil its obligations, the Company intends to opt for a financial settlement, but is interested in investigating a proponent driven offset (at least in part) involving the rehabilitation of Cabbage Tree Creek utilizing an indigenous contractor.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility and Infrastructure

The Project tenements are in North West Queensland and are shown in Figure 4-1. Access to the Project is by the sealed Barkly Highway from Mount Isa to Cloncurry, then on the sealed Burke Developmental Road, through Quamby. The highway passes 8.5 km to the east of the proposed plant site, and current access is by way of a gravel road. The planned site for the plant and major infrastructure is 11 km due north of the Dugald River zinc mine owned by MMG, which had first production in November 2017.

The Project is located about 65 km (76 km by road) northwest of Cloncurry, a town of about 3,000 inhabitants, and about 95 km (194 km by road) from Mount Isa, a regional mining centre with a population of about 22,000 people.

Cloncurry is located on the railway line from Townsville to Mount Isa and has container handling facilities, an airport (which hosts both commercial and fly-in/fly-out (FIFO) jet aircraft services), and a regional fuel depot. Cloncurry also has schools, hospitals, and other services. The Project lies within the Shire of Cloncurry local government administrative area and the Shire offices are based in Cloncurry.

Grid power is reticulated from Mount Isa to Cloncurry, and power is generated in Mount Isa at two gas-fired power stations. A 220-kV power line has been constructed from the Chumvale substation near Cloncurry to the Dugald River mine, 11 km from the proposed Eva Copper Project process plant.

Quamby is a tiny hamlet to the southeast of the proposed plant site, with a now-closed roadhouse on the highway, and a Telstra communications tower.

Kajabbi is a small hamlet to the north of the area and has stockyards that were used for loading cattle onto a railway line that used to run south through Quamby to Cloncurry. The railway line from Cloncurry north to Kajabbi has been removed, and all that remains is the easement, which is still owned by Queensland Transport.

A water pipeline operated by SunWater passes within 2 km of the Eva Copper Project plant site, and is fed from Lake Julius, 41 km to the west, and reticulated to the Ernest Henry mine, and the Cloncurry townsite. Dugald River also has a water take-off from this pipeline. The pipeline has a capacity of seven gigalitres per year (GL/a).

5.2 Climate and Surface Water

The Bureau of Meteorology weather station closest to the Project site is located on McIlwraith Street, Cloncurry, and has records dating back to 1884. The mean annual maximum temperature is 32.2°C, and the average annual rainfall for the region is 474 mm/a.

Mean temperatures in the dry season range from 26.2°C to 36.4°C from April to October.

Temperatures range from mean monthly highs of 26.2°C to 38.5°C, to monthly lows of 10.6°C to 24.8°C. Minimum and maximum recorded temperatures range from to 1.8°C to 46.9°C. The hottest months correspond with the wet season, between November and March.

Mean wind speeds measured at the Mount Isa Airport weather station shows that the later months of the year exhibit the highest wind speeds, peaking in October at an average speed of 15.8 km/h. Wind speeds are lowest in the cooler months of the year, at an average of 9.5 km/h in June. Maximum wind gusts range from a low of 63 km/h in July up to 128 km/h in January.

The relative humidity at the Cloncurry weather station peaks in February, typically reaching 39% at 3:00 p.m., and 61% at 9:00 a.m. The peak fire season for the Project area is winter to spring (July to September), when the vegetation is at its driest.

Rainfall is seasonal, largely occurring between November and March (wet season), and generally occurs in large storms. Rainfall is highly variable from year to year, with the region often experiencing multi-year droughts and large-scale flooding from major rainfall events.

The Project site is serviced by a complex system of surface drainages that flow generally northward. On the western side of the plant and Little Eva pit is Cabbage Tree Creek, which is joined by other creeks flowing northward to become a tributary of the Leichhardt River. The central parts of the Mining Leases (ML) drain into the Dugald River. Numerous other minor ephemeral watercourses cross the Project area.

Creeks and rivers only flow during, and for a brief period following, the wet season. Intensive rains, with cumulative falls up to 50 mm over a few days, generate flows in the larger creeks, such as Cabbage Tree Creek and Dugald River. Peak flows are generally of short duration. Most stream flow ceases within days or a few weeks after intensive wet periods, after which the flow channel breaks into isolated pools. The rivers and creeks have a composite profile consisting of a steep-sided main channel 1 m to 1.5 m in depth in which flows occur annually, often to bank height. Isolated pools in the riverbeds can persist through the dry season in sand, gravel, and crystalline rock fractures. Water can generally be found below the riverbeds at a depth of one to two metres.

After the dry season, storm rains of approximately 25 mm/d may occur, which may include intense periods equivalent to 24 mm/h, which would generate runoff in the smaller creeks.

The Project has groundwater sources from both hard rock fracture zone systems and from a graben-like structure filled in with Phanerozoic sediments. In addition to this geological feature, the main creeks are associated with extensive thin sheets of colluvial outwash and alluvial deposits, with groundwater present in the deeper parts of these deposits.

5.3 Landforms and Vegetation

The Project site and broader operation area is gently undulating, with the Knapdale range of hills rising quite sharply from the plain to the south of the proposed operations area, with a length of approximately 12 km, and rising to an average height of 300 Australian Hight Datum metres above sea level (mASL). A discrete north–south ridgeline, which includes Mount Rose Bee and the Green Hills, transects the area on the western side of the Bedford deposit. Mount Rose Bee (approximately 285 mASL) is characterized by ridges of exposed silicified rock.

The site is currently crossed by several access tracks from farming and exploration activities. SunWater's water pipeline from Lake Julius to the Ernest Henry mine crosses the lease area from west to east.

The predominant land use is low-intensity cattle grazing, although exploration and mining activities have been conducted over the area since the late 1800s. Soils of the Project site are typically slightly acid to moderately alkaline, and non-sodic and therefore non-dispersive in nature, meaning they are not chemically predisposed to erosion. Most of the erosion potential of these soils originates from the short duration, high intensity rainfall events that can occur during the summer period (December to March).

5.4 Local Mining Industry

Mount Isa was established on the discovery of world-scale copper-zinc-lead deposits in 1923. A major mining complex and a town of 22,000 people has grown on the site in the last 94 years, with multiple open pit and underground mines, smelters, mills, flotation plants, and a sulphuric acid plant. The town of Mount Isa hosts many mining suppliers, service organizations, and a number of skilled mining industry people, as well as having two electric-powered generators supplied by a natural gas pipeline from South Australia, an airport, rail line, and other services.

Cloncurry was established much earlier than Mount Isa, on the discovery of copper by Ernest Henry in 1867, and the town was founded in 1884.

There are numerous active mines in the area, as shown in Figure 5-1. In addition to Mount Isa, there are five major active mines: the Ernest Henry copper-gold mine and Lady Loretta lead-zinc-silver mine, both owned by Glencore; the Cannington silver-lead mine owned by South 32; the Dugald River zinc-lead-silver mine owned by MMG; and the Capricorn Copper copper-gold mine owned by Capricorn Copper. All are major, internationally important mines.

Smaller operations (active and in care and maintenance) include: Osborne copper-gold mine, owned by Chinova; Mount Colin copper mine, owned by Round Oak Minerals, Lady Annie copper-gold mine, owned by CST Mining; Mount Cuthbert Copper mine, owned by Malaco Mining; Rocklands copper-gold mine, owned by Cudeco; and Eloise copper-gold mine, owned by FMR Investments.

Closed major mines include the Mary Kathleen uranium mine.

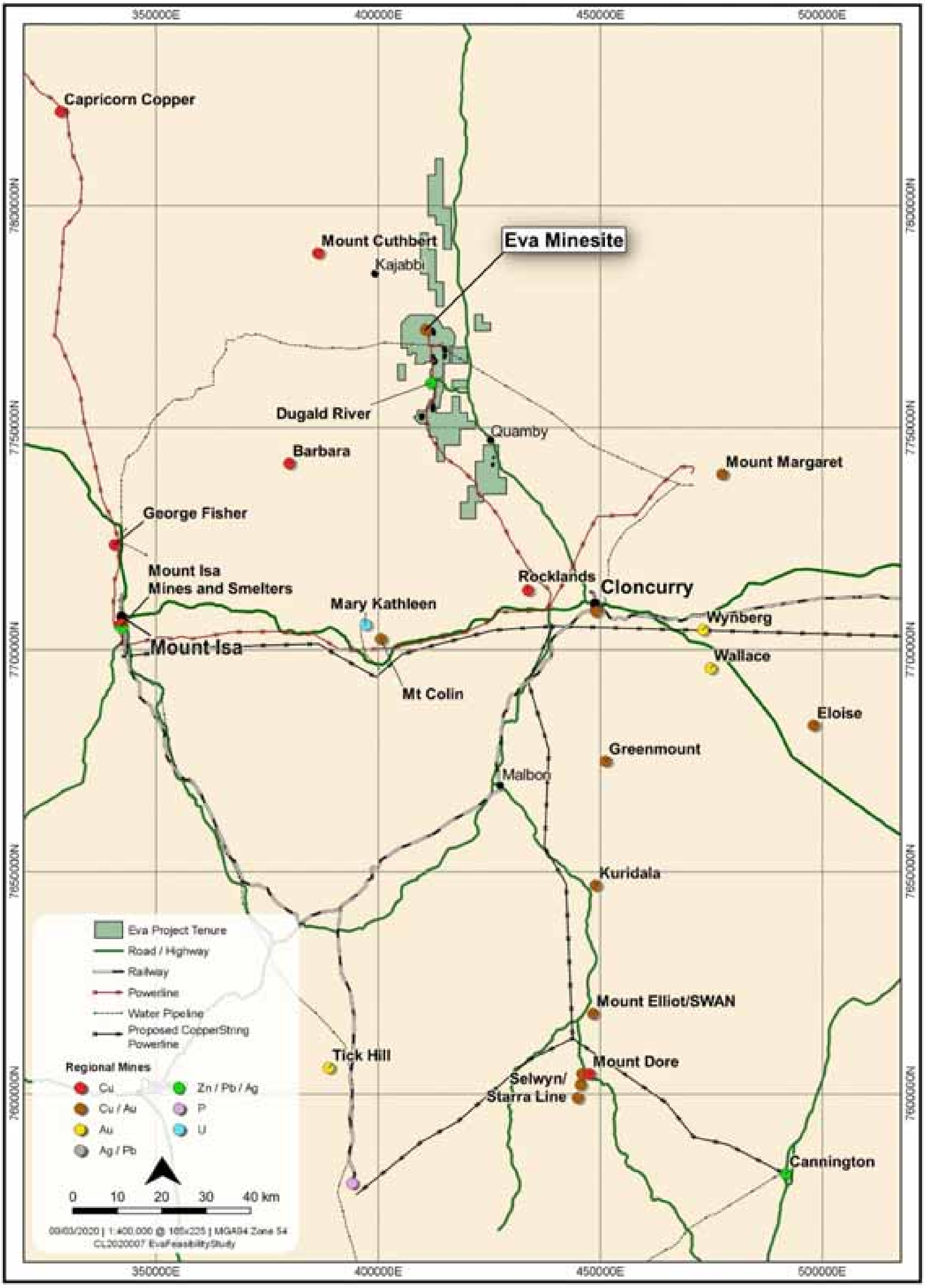


Figure 5-1: Infrastructure, Major Mines, Deposits, and Eva Copper Project Tenure

6 HISTORY

6.1 Prior Ownership and Changes

The Project area has a long history of exploration and development. Early work was undertaken by Ausminda Pty. Ltd. and CRA Exploration (CRAE) between 1990 and 1996. CRAE's principal focus was the copper-only deposits where they were successful in discovering a number of deposits. The Little Eva and Lady Clayre deposits were of secondary interest to CRAE, who drilled the Little Eva deposit to define a small deposit of 9 Mt assayed at 0.70% copper (Cu), gold (Au) grade was not reported.

In 1996, the property was acquired by Pasminco Limited (Pasminco), who undertook further exploration and drilling on the copper-only deposits. Pasminco excised and retained the Dugald River zinc deposit, and sold the remainder of the tenements to Universal in 2001. The Little Eva deposit was first fully delineated by Universal. Pasminco was taken over by Zinifex in 2002, and in 2008 Zinifex merged with Oxiana to become Oz Minerals. Oz Minerals' interest in the Dugald River zinc deposit was acquired in 2009 by MMG, a subsidiary of China Minmetals.

From 2001 to 2004, exploration work on the Blackard, Scanlan, and Longamundi copper-only deposits was carried out under a joint venture (JV) between Universal and Bolnisi Logistics. In 2004, Universal acquired Bolnisi Logistics and assumed full management of the Project. Bolnisi Logistics then changed its name to Roseby Copper Pty. Ltd. Universal focused its 2001–2004 drilling on the Little Eva and Bedford copper-gold deposits, and completed a Feasibility Study in 2005 based on mining and processing a blend of sulphide ore from the Little Eva and Bedford deposits with native copper ore from the Blackard and Scanlan deposits; however, Universal did not proceed with development.

Universal entered into a JV Option Agreement with Xstrata in 2005, where Xstrata had the right to explore in the central area of the tenements. Xstrata discovered the Cabbage Tree Creek prospect, and significant sulphide mineralization beneath the Blackard deposit. Xstrata elected not to proceed with the option to purchase an interest in the Project in January 2013. Universal completed a second Feasibility Study between 2007 and 2009 based on the same blend of sulphide ore and native copper ore used in the 2005 study.

In December 2009, Universal merged with Vulcan Resources Limited, and the company name changed to Altona Mining Limited (Altona). Altona drilled out the Little Eva deposit, doubling the Mineral Resource, and in 2012 completed a Definitive Feasibility Study (DFS) based on the increased resources of copper-gold sulphide deposits, with this report excluding the Blackard and Scanlan deposits. Altona's philosophy was to take a simpler approach that did not rely on ore blending and to address mining and processing of native copper ores once operations were established, in the context of extending mine life or increasing the production rate.

Altona completed drilling at the Bedford, Lady Clayre, Ivy Ann, Blackard, Legend, and Scanlan deposits, and published Mineral Resource upgrades for all these deposits. Altona published Mineral Reserves for the Little Eva, Bedford, Lady Clayre, and Ivy Ann deposits as part of their 2012 DFS. Altona discovered a significant resource at Turkey Creek and published Mineral Resource and Mineral Reserve estimates for the deposit in 2015 and 2016, respectively. Altona also discovered and delineated major prospects at Anzac, Whitcher, Matchbox, and Quamby from 2015 to 2016.

Mining Leases (ML) and an EA were granted in 2012 based on the 2009 DFS mine plan. An EA amendment was granted in 2016 based on the revised 2012 DFS mine plan and the integration of Turkey Creek into that mine plan.

Prior to the acquisition of Altona by Copper Mountain Mining Corp. (CMMC), and excluding acquisition costs, approximately $63 million has been expended on exploration, resource development, metallurgical and engineering studies, compensation payments, and government fees and charges by the various parties involved over the past 27 years, including:

- CRAE (estimate) .... $7.4 million
- Zinifex/Pasminco................. $0.7 million
- Bolnisi.................................. $4.1 million
- Universal .... $24.2 million
- Xstrata .... $8.5 million
- Altona .... $18.4 million

Note: Exchange rate AU$1.35:US$1.00

On April 4, 2018, Altona became a wholly owned Australian subsidiary of CMMC, and was subsequently renamed CMMPL.

6.2 Mineral Resource Estimates History

6.2.1 Little Eva Deposit

The Little Eva deposit has had several formal Mineral Resource estimates that reflect stages of resource definition (Table 6-1). The 2008 and 2012 Mineral Resource estimates include both sulphide and oxide material, while the 2014 estimate is for sulphide material only, with the oxide material excluded.

Table 6-1: Little Eva Resource Estimate History

Model	Authors	Mineral Resource Estimate	Comment
Oct 2008*	MacDonald Speijers	30.4 Mt at 0.78% Cu, 0.09 g/t Au. (0.3% Cu cut-off grade).	Superseded following additional drilling Includes Inferred resource.
Mar 2012*	Optiro and Altona	108 Mt at 0.52% Cu, 0.9 g/t Au. Sulphide mineralization – 100.3 Mt at 0.53% Cu and 0.09 g/t Au at a 0.2% Cu cut-off grade.	Basis for 2012 DFS and Mineral Reserve estimation. Primary sulphide and oxide mineralization. Includes Inferred resource.
May 2014*	Altona and Optiro	105.9 Mt at 0.52% Cu, 0.09 g/t Au at a 0.2% Cu cut-off grade.	Sulphide mineralization only. Includes Inferred resource.
Nov 2018	CMMC	121.8 Mt at 0.36% Cu, 0.07 g/t Au at a 0.17% Cu cut-off grade.	Basis for this study. Nominal additional infill drill data only. Sulphide mineralization only. Excludes Inferred resource.

Source: *Altona Mining Limited, Cloncurry Copper Project – DFS, August 2017.
Total estimated Mineral Resource including Inferred; reported in accordance with the Joint Ore Reserves Code (JORC).

In October 2008, McDonald Speijers completed a Mineral Resource estimate for the Little Eva deposit, reported in accordance with JORC, 2004 Edition (JORC, 2004), which was incorporated into Universal's 2009 Roseby Copper Project Feasibility Study. The Mineral Resource amounted to 30.4 Mt at 0.78% Cu and 0.9 g/t Au, with a 0.3% Cu cut-off grade. Geological domains were poorly constrained.

In March 2012, Altona, in conjunction with Optiro, reported a 2004 JORC-compliant Mineral Resource. This estimate incorporated newly acquired assay and geological data provided by extensive drill programs. The published estimate of 108 Mt at 0.52% Cu and 0.09 g/t Au includes both sulphide and oxide mineralization. This estimate formed the basis of pit optimizations used in the 2012 DFS and 2014 DFS update.

In May 2014, Altona published a revised Mineral Resource estimate, which was reported and classified in accordance with JORC (JORC, 2012). Geological models were limited in former estimates, because of inconsistent drill hole logging between multiple corporations and programs. The 2014 used a new geological model based on a detailed drill hole relogging program by Altona in 2013-2014. The estimate of 105.9 Mt at 0.52% Cu and 0.09 g/t Au is for primary sulphide geotechnical mineralization only, and excludes oxide mineralization, which is not amenable to processing through the proposed Eva Copper Project plant. No additional drilling was added after the 2012 estimate leading up to the 2014 estimate, and differences between the 2012 and 2014 Mineral Resource estimates were not considered material; however, confidence in the geological models was improved. The 2014 estimate was used for pit optimizations undertaken by Orelogy, and to relocate mine and waste dump layouts and develop schedules to accommodate Turkey Creek into the mine plan. This work was undertaken for the 2016 EA Amendment. These optimizations were not used in the mine design or financial modelling of the August 2017 Altona DFS.

In November 2018, CMMC published the current Mineral Resource estimate, which was reported and classified in accordance with JORC 2012 and CIM 2014. This estimate incorporated limited newly acquired assay and geological data provided by diamond core holes drilled for metallurgical, geotechnical, and due diligence purposes. The published estimate of 121.8 Mt at 0.36% Cu and 0.07 g/t Au includes sulphide mineralization only. This estimate forms the basis of pit optimizations used in CMMC's 2018 Feasibility Study and this Feasibility Study update.

Limited additional drilling was added after the 2014 estimate; differences between the 2014 and 2018 Mineral Resource estimates reflect a lower minimum reporting cut-off grade, different modelling approach, and exclusion of material classified as Inferred from the reported total.

6.2.2 Turkey Creek Deposit

Turkey Creek was discovered in September 2012 after the 2012 DFS was completed. The only Mineral Resource estimate for Turkey Creek was completed in 2015 by Optiro and Altona. Altona was responsible for the data and 3D geological model. Mineral Resource estimation and block modelling was conducted by Optiro. The Mineral Resource estimate was classified and reported in accordance with JORC 2012. The estimate was 21 Mt grading 0.59% Cu, and it includes both sulphide and oxide mineralization.

Table 6-2: Turkey Creek Resource Estimate History

Model	Authors	Mineral Resource Estimate	Comment
Mar 2015*	Optiro and Altona	21 Mt at 0.59% Cu (0.3% Cu cut-off grade)	Primary sulphide and oxide mineralization. Include Inferred resources.
Nov 2018	CMMC	13.8 Mt at 0.46% Cu (0.17% Cu cut-off grade)	Basis for this study. Nominal additional infill diamond drill data only. Primary sulphide mineralization only. Excludes Inferred resources.

Source: *Courtesy Altona Mining Limited, Cloncurry Copper Project – Definitive Feasibility Study, August 2017. Total estimated Mineral Resource including Inferred; reported in accordance with JORC.

In November 2018, CMMC published the current Mineral Resource estimate, which was reported and classified in accordance with JORC 2012 and CIM 2014. This estimate incorporated limited newly acquired assay and geological data provided by diamond core holes drilled for metallurgical, geotechnical, and due diligence purposes. The published estimate of 13.8 Mt at 0.46% Cu includes sulphide mineralization only. This estimate forms the basis of pit optimizations used in CMMC's 2018 Feasibility Study and this Feasibility Study update.

The 2015 Mineral Resource for Turkey Creek included an oxide component, while the other deposits modelled did not. There was a reasonable expectation of achieving acceptable recoveries from the oxide material based on the mineralogy using the Controlled Potential Sulphidization (CPS) technique for flotation processing. However, initial metallurgical testing of this processing method produced poor recoveries, and the oxide material was excluded from the 2018 Mineral Resource.

The 2015 resource model was used by Orelogy to generate pit designs and waste volumes included in the mine plan, and was used to generate a new layout of pits, waste dumps, and the tailings storage facility (TSF) for the 2016 EA. The Orelogy pit optimizations were based on primary sulphide mineralization only. The sulphide resource was estimated at 16.5 Mt at 0.59% Cu.

Limited additional drilling was added after the 2015 estimate, and differences between the 2015 and 2018 Mineral Resource estimates reflect a lower minimum reporting cut-off grade, different modelling approach, and exclusion of material classified as Inferred from the reported total.

6.2.3 Bedford Deposit

The Bedford deposit has had several formal Mineral Resource estimates completed that reflect stages of resource definition, as shown in Table 6-3. The 2012 and 2017 estimates are for sulphide material only.

In October 2006, McDonald Speijers completed an initial Mineral Resource estimate. In May 2012, Optiro completed an independent estimate of recoverable resources based on nominal additional drilling. There was no significant change in the Mineral Resource of 1.7 Mt at 0.99% Cu and 0.20 g/t Au, and both estimates were reported in accordance with JORC 2004.

Geological models forming the basis of these estimates were poorly constrained, with isolated individual structures within a broader shear zone showing limited continuity. The 2012 estimate by Optiro forms the basis of pit optimizations and financial models used in the 2012 DFS and the 2014 DFS update.

Table 6-3: Bedford Resource Estimate History

Model	Authors	Mineral Resource Estimate	Comment
Oct 2006*	McDonald Speijers	1.77 Mt at 0.93% Cu, 0.24 g/t Au (0.3% Cu cut-off grade)	Superseded following nominal additional drilling. Includes Inferred resource.
May 2012*	Optiro	1.7 Mt at 0.99% Cu, 0.20 g/t Au (0.3% Cu cut-off grade)	Basis for 2012 DFS and Mineral Reserve estimate. Primary sulphide mineralization only. Includes Inferred resource.
Feb 2017*	Altona	4.8 Mt at 0.80% Cu, 0.21 g/t Au (0.3% Cu cut-off grade)	Assay data from two additional drill holes. Additional geological data showing continuity of structures. Primary sulphide mineralization only. Includes Inferred resource.
Nov 2018	CMMC	3.0 Mt at 0.54% Cu and 0.14 g/t Au (0.17% Cu cut-off grade)	Basis for this study. Primary sulphide mineralization only. Excludes Inferred resource.

Source: Altona Mining Limited, Cloncurry Copper Project – Definitive Feasibility Study, August 2017.
*Total estimated Mineral Resource including Inferred; reported in accordance with JORC.

In February 2017, Altona completed a new (and current) Mineral Resource estimate, which was reported and classified in accordance with JORC 2012. The estimate of 4.8 Mt at a grade of 0.80% Cu and 0.21 g/t Au includes primary sulphide mineralization only. The increase from the 2012 estimate resulted primarily from a better understanding of geological continuity and geometry. Mineralized structures are better defined by mapping of surface workings and high-resolution copper-in-soil sampling. An increase in tonnage was a result of more accurate bulk density data obtained from diamond drill core, therefore replacing prior bulk density estimates.

In November 2018, CMMC published the current Mineral Resource estimate, which was reported and classified in accordance with JORC 2012 and CIM. The published estimate of 3.0 Mt at 0.54% Cu and 0.14 g/t Au includes sulphide mineralization only. This estimate forms the basis of pit optimizations used in CMMC's 2018 Feasibility Study and this Feasibility Study update.

No significant new drill data was added after the 2017 estimate, and differences between the 2017 and 2018 Mineral Resource estimates reflect a lower minimum reporting cut-off grade, different modelling approach, and exclusion of material classified as Inferred from the reported total.

6.2.4 Lady Clayre Deposit

The Lady Clayre deposit has had several formal Mineral Resource estimates that reflect stages of resource definition, as shown in Table 6-4.

In October 2006, McDonald Speijers completed a Mineral Resource estimate that was reported in accordance with JORC 2004 for the Lady Clayre deposit. This was incorporated into Universal's 2009 Feasibility Study.

Table 6-4: Lady Clayre Resource Estimate History

Model	Authors	Mineral Resource Estimate	Comment
Oct 2006*	McDonald Speijers	3.7 Mt grading 0.88% Cu, 0.48 g/t Au (0.3% Cu cut-off grade	Superseded following expanded drilling.
May 2012*	Optiro	14 Mt grading 0.56% Cu, 0.20 g/t Au (0.3% Cu cut-off grade	Superseded by this Feasibility Study Mineral Resource estimate. Basis for 2012 DFS and Mineral Reserve estimate. Primary sulphide mineralization only. Includes Inferred resource.
Nov 2018	CMMC	7.3 Mt at 0.31% Cu and 0.14 g/t Au (0.17% Cu cut-off grade)	Basis for this study. Primary sulphide mineralization only. Excludes Inferred resource.

Source: Courtesy Altona Mining Limited, Cloncurry Copper Project – Definitive Feasibility Study, August 2017.
*Total estimated Mineral Resource including Inferred; reported in accordance with JORC.

In November 2018, CMMC published the current Mineral Resource estimate, which was reported and classified in accordance with JORC 2012 and CIM. The published estimate of 7.3 Mt at 0.31% Cu and 0.14 g/t Au includes sulphide mineralization only. This estimate forms the basis of pit optimizations used in CMMC's 2018 Feasibility Study and this Feasibility Study update.

Significant new drill data was added after the 2012 estimate; the updated model used the new drill hole assay data but was not constrained by the revised geological model. Differences between the 2012 and 2018 Mineral Resource estimates reflect new drilling data, a lower minimum reporting cut-off grade, different modelling approach, and exclusion of material classified as Inferred from the reported total.

6.2.5 Ivy Ann Deposit

The Ivy Ann deposit has had three Mineral Resource estimates, as shown in Table 6-5.

Table 6-5: Ivy Ann Resource Estimate History

Model	Authors	Mineral Resource Estimate	Comment
Jan 2006*	Universal	3.98 Mt at 0.93% Cu, 0.24 g/t Au (0.3% Cu cut-off grade)	Superseded following expanded drilling.
May 2012*	Optiro	7.5 Mt at 0.57% Cu, 0.07 g/t Au (0.3% Cu cut-off grade)	Basis for 2012 DFS Mineral Reserve estimate. Primary sulphide mineralization only. Includes Inferred resource.
Nov 2018	CMMC	5.1 Mt at 0.36% Cu and 0.08 g/t Au (0.17% Cu cut-off grade)	Basis for this study. Primary sulphide mineralization only. Excludes Inferred resource.

Source: Altona Mining Limited, Cloncurry Copper Project – Definitive Feasibility Study, August 2017.
*Total estimated Mineral Resource including Inferred; reported in accordance with JORC.

In January 2006, Universal completed a Mineral Resource estimate for the Ivy Ann deposit, which was reported in accordance with JORC 2004. This estimate was not incorporated into Universal's 2009 Feasibility Study.

In May 2012, Optiro completed an estimate of resources for the Ivy Ann deposit that incorporated additional drilling. The published estimate of 7.5 Mt at 0.57% Cu, and 0.07 g/t Au includes sulphide mineralization only. The estimate was used for optimization and Mineral Reserve estimation for the 2012 DFS.

In November 2018, CMMC published the current Mineral Resource estimate, which was reported and classified in accordance with JORC 2012 and CIM. The published estimate of 5.1 Mt at 0.36% Cu and 0.08 g/t Au includes sulphide mineralization only. This estimate forms the basis of pit optimizations used in the CMMC 2018 Feasibility Study and this Feasibility Study update.

No significant new drill data was added after the 2012 estimate, and differences between the 2012 and 2018 Mineral Resource estimates reflects a lower minimum reporting cut-off grade, different modelling approach, and exclusion of material classified as Inferred from the reported total.

6.2.6 Blackard Deposit

The Blackard deposit has had several formal Mineral Resource estimates that reflect stages of resource definition, as shown in Table 6-6. While early estimates from 2003 included all mineralization (oxide, copper, transition, and sulphide zones) the 2012 Mineral Resource estimate only includes native copper, transition and primary sulphide, the oxide zone was excluded.

In December 2005, McDonald Speijers completed a Mineral Resource estimate that was reported in accordance with the JORC 2004 for the Blackard deposit; this was incorporated into Universal's 2006 Feasibility Study. In February 2007, McDonald Speijers completed a Mineral Resource estimate update; this was incorporated into Universal's 2009 Feasibility Study. In both cases the estimates were used for pit optimizations with resultant Ore Reserve estimates.

In May 2012, Optiro completed an independent estimate of recoverable resources for the Blackard deposit. The estimate of 76.4 Mt at 0.62% Cu includes native copper, transition, and primary sulphide mineralization only. This estimate incorporated newly acquired data from substantial additional drilling since the 2006 estimate. The deposit was not considered in Altona's 2012 to 2017 Feasibility Studies and updates, which focused on demonstrating Project viability based on mining and processing sulphide mineralization only.

Table 6-6: Blackard Resource Estimate History

Model	Authors	Mineral Resource Estimate	Comment
May 1996	Newbery and Lai (for CRAE)	27 Mt at 0.73% Cu (0.5% Cu cut-off grade)	Superseded by new model for Bolnisi. Oxide (malachite), native copper, and transition mineralization only. Indicated resource only.
Feb 2003*	Hellman & Schofield	26.8 Mt at 0.75% Cu (0.5% Cu cut-off grade)	Superseded following expanded drilling. Oxide (malachite), native copper, transition, and primary sulphide mineralization. Includes Inferred resource.
Dec 2005*	McDonald Speijers	43.7 Mt at 0.65% Cu (0.3% Cu cut-off grade)	Superseded following expanded drilling. Oxide (malachite), native copper, transition, mineralization only. Includes Inferred resource.
Jan 2007*	McDonald Speijers	46.25 Mt at 0.63% Cu (0.3% Cu cut-off grade)	Superseded following expanded drilling. Oxide (malachite), native copper, transition, and primary sulphide mineralization. Includes Inferred resource.
Jul 2012*	Optiro	76.4 Mt at 0.62% Cu (0.3% Cu cut-off grade)	Superseded by this Feasibility Study Mineral Resource estimate. Native copper, transition, and primary sulphide mineralization only. Includes Inferred resource.
Oct 2019	CMMC	77.3 Mt at 0.49% Cu (0.23% Cu, 0.20% Cu, and 0.17% Cu cut-off grade for copper, transition, and sulphide zone, respectively)	Basis for this study. Native copper, transition, and primary sulphide mineralization only. Excludes Inferred resource.

Source: Altona Library, resource estimation reports for Altona and previous Project operators.
*Total estimated Mineral Resource including Inferred; reported in accordance with JORC.

Data from 18 new drill holes was added to the 2019 resource estimate. Differences between the reported 2012 and 2019 Mineral Resource estimates reflect a lower cut-off grades, different modelling approach, and exclusion of Inferred resources.

6.2.7 Scanlan Deposit

The Scanlan deposit has had three Mineral Resource estimates, as shown in Table 6-7.

In November 2006, McDonald Speijers completed a Mineral Resource estimate that was prepared in accordance with the JORC 2004 for the Scanlan deposit. This was incorporated into Universal's 2006 and 2009 Feasibility Studies. In both cases the estimates were used for pit optimizations with resultant Ore Reserve estimates.

Table 6-7: Scanlan Resource Estimate History

Model	Authors	Mineral Resource Estimate	Comment
May 1995	Newbery and Lai (for CRAE)	15 Mt at 0.81% Cu (0.5% Cu cut-off grade)	Superseded by new model for Bolnisi. Oxide (malachite), native copper, and transition mineralization only. Indicated resource only.
Nov 2006*	McDonald Speijers	19.62 Mt at 0.68% Cu (0.3% Cu cut-off grade)	Superseded following additional drilling.
Jul 2012*	Optiro	22.2 Mt at 0.65% Cu (0.3% Cu cut-off grade)	Native copper, transition, and primary sulphide mineralization only. Includes Inferred resource.
Jan 2020	CMMC	21.7 Mt at 0.57% Cu (0.26% Cu, 0.20% Cu, and 0.17% Cu cut-off grade for copper transition and sulphide zones, respectively)	New this this study. Excludes Inferred resource.

Source: Altona Library, resource estimation reports for Altona and previous Project operators.
Total estimated Mineral Resource including Inferred; reported in accordance with JORC.

In July 2012, Optiro completed an independent estimate of recoverable resources for the Scanlan deposit. The estimate of 22.2 Mt at 0.65% Cu includes native copper, transition, and primary sulphide mineralization only. This estimate incorporated newly acquired data from substantial additional drilling completed since the 2006 estimate. The deposit was not considered in Altona's 2012 to 2017 Feasibility Studies and updates, which focused on demonstrating Project viability based on mining and processing sulphide mineralization only.

No significant drill data has been added to the deposit since 2012.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Project is located within the Proterozoic rocks of the Mount Isa Province of Queensland, Australia. The region is one of the world's premier base metal provinces, with mining continuing uninterrupted since discovery of copper and gold near Cloncurry in the 1860s. The Mount Isa Province hosts numerous copper mines, including several of global significance. The Mount Isa Province also hosts the world's two largest lead producers, the second largest silver producer, and until recently was the world leading source of zinc. Economic accumulations of various other commodities, including gold, molybdenum, rare earth elements, uranium, and phosphate, occur throughout the area.

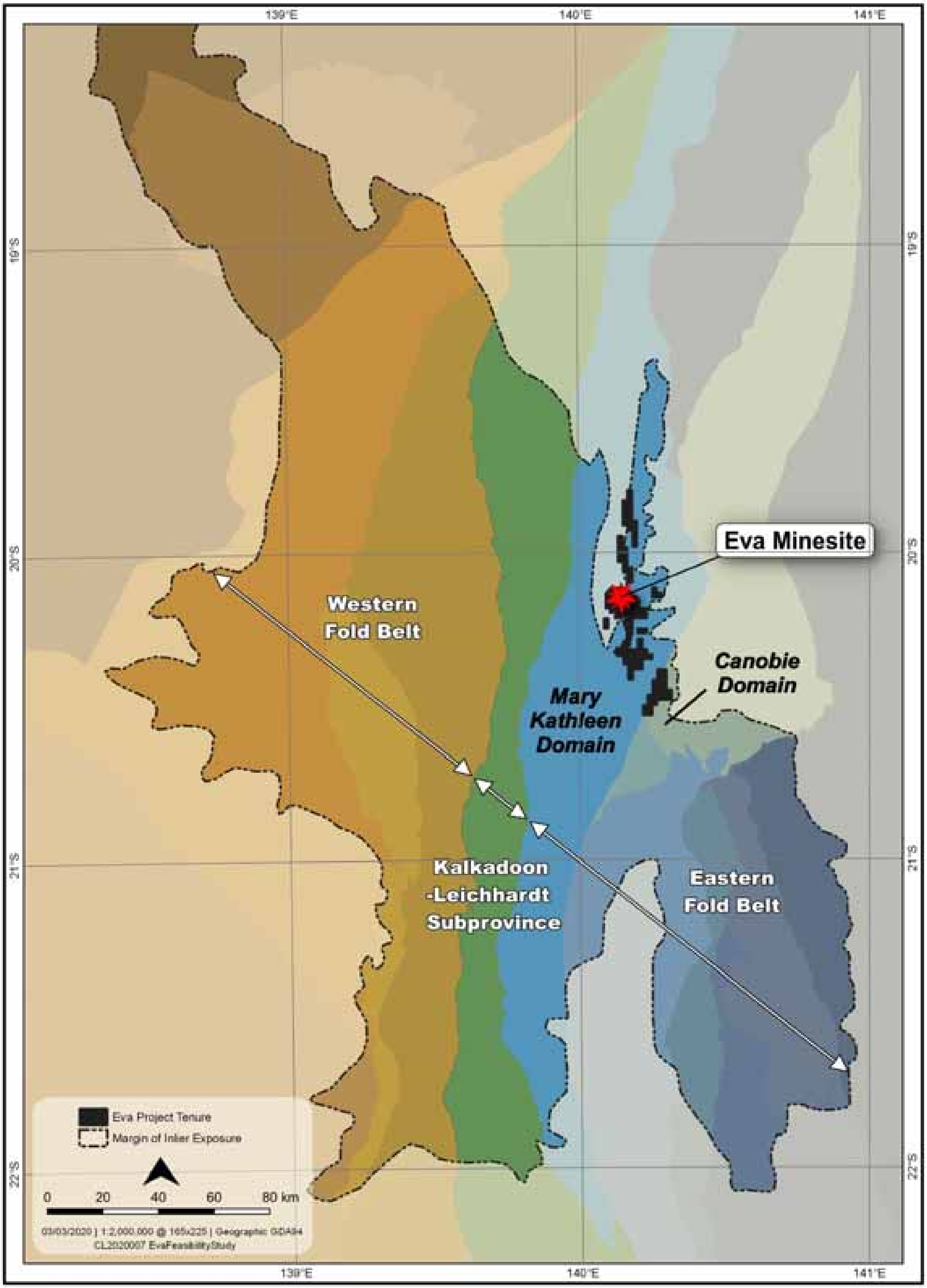
The Project is situated within the Mary Kathleen domain, and to a lesser extent the Canobie domain of the late Palaeoproterozoic Eastern Fold Belt of the Mount Isa Inlier (Figure 7-1), which largely comprises metamorphosed marine sedimentary and volcanic rocks some 1,590 to 1,790 Ma old. Numerous granite and mafic intrusions were emplaced at various times before 1,100 Ma.

The Project area rocks have undergone polyphase deformation, metamorphism, and metasomatism during the Isan Orogeny (1,600–1,500 Ma), which resulted in east-west shortening and extensive plutonism. The orogeny formed the major north-south trending upright folds and structural domains that characterize the province. Deformation and late- to post-orogenic plutonism is most pronounced in the Eastern Fold Belt where it is associated with widespread high temperature sodium-iron metasomatism expressed as magnetite or hematite alteration assemblages. Iron-oxide-copper gold (IOCG) mineralization is a variant of the Na-Fe metasomatism and the Project deposits are examples of such mineralization. IOCG mineralization developed in the waning stages of the Isan Orogeny, and is prevalent throughout the Eastern Fold Belt.

North- and north-easterly-trending crustal scale faulting transects the Province, bounding and cutting geological domains. The structures are the locus of major base and precious metal deposits. The deformation recorded by faulting and folding is complex, dominated at different stages by extension, shortening, and transcurrent faulting. The major faults have long reactivation histories during the Proterozoic, with evidence of recurrent activity in the Phanerozoic. During the latter part of the Isan Orogeny, at the time of IOCG mineralization, the pre-existing faults were reactivated into a dominantly strike-slip wrench system, with east-west to southeast-northwest directed shortening accompanying emplacement of the William Batholiths (1,530–1,490 Ma).

The Project deposits are located within the Mary Kathleen (MK) domain, which is an elongate belt on the east side of the Kalkadoon-Leichhardt domain, has a length of 180 km and an approximate width of 20 km, and was modified by the Wonga extensional event (approx. 1,740 Ma) which included emplacement of the Wonga Suite granites. The MK domain hosts the Dugald River zinc deposit, the Tick Hill gold deposit, the Mary Kathleen uranium deposit, and the Phosphate Hill phosphate deposit, in addition to the Project's copper-gold deposits.

The Canobie domain is located east of the Mary Kathleen domain, and the two are juxtaposed by the Fountain Range and Pilgrim Faults. The Canobie domain is fault bounded, poorly exposed, largely defined by highly magnetic and buried William-Naraku intrusions and is host to the Ernest Henry copper-gold deposit.



Note: Little Eva deposit denoted by red star

Figure 7-1: Geological Domains of the Mount Isa Province and Project Location

7.1.1 Regional Stratigraphy

The Little Eva deposit, which contains most of the resources in the Eva Copper Project, lies within the northern exposed portion of the Mount Isa Eastern Succession. Rocks within this area include a variety of Palaeoproterozoic sediments and volcanic and intrusive rocks, as illustrated in Figure 7-2 and Figure 7-3. The Palaeorproterozoic age (1,770±5 Ma) Corella Formation dominates the deposit area, and comprises scapolitic calcareous metasediments, quartzites, and granofels (Betts et al., 2011).

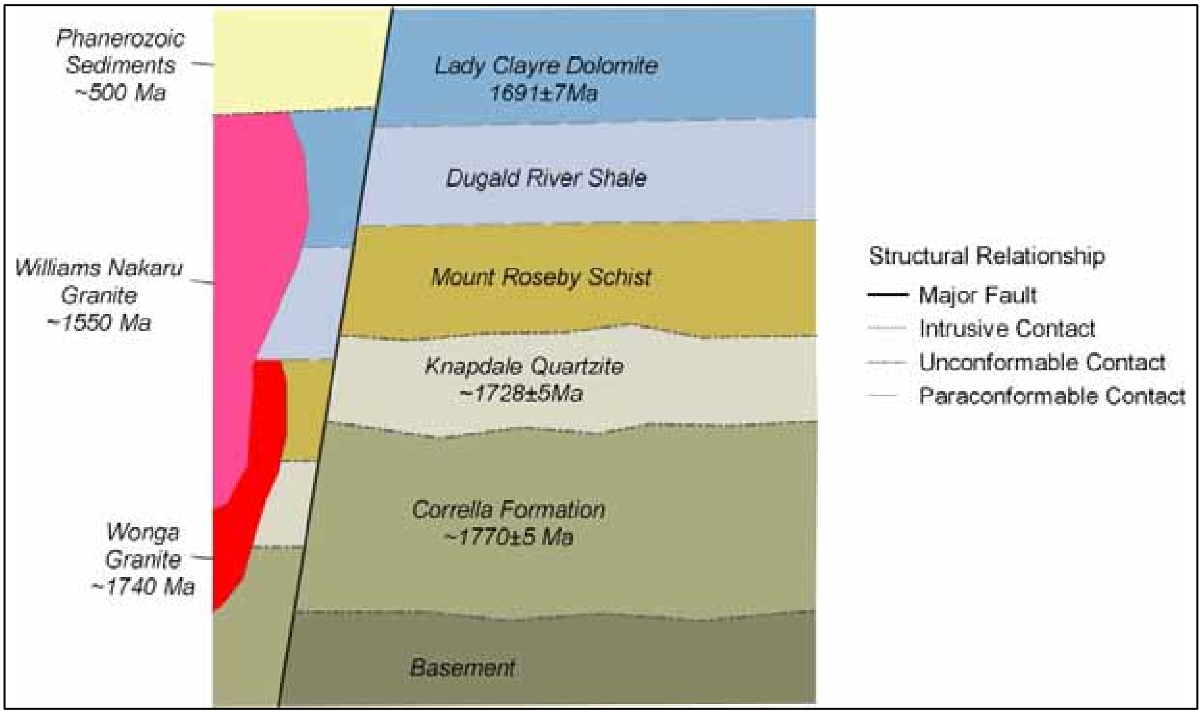


Figure 7-2: Schematic Stratigraphic Diagram of the Little Eva Deposit Area

Approximately 1,740 Ma, deposition of the Mount Isa eastern succession was terminated by a period of significant extension referred to as the Wonga Event. The Wonga Event was accompanied by dominantly felsic extrusive and intrusive magmatism (Greenwood & Dhnaram, 2013). Sedimentation resumed following the Wonga Event, with deposition of the Knapdale quartzite (feldspathic and micaceous sandstone and quartzite) at 1,728±5 Ma (Greenwood & Dhnaram, 2013; Betts et al., 2012). Additional sedimentation occurred with deposition of material that would become the Mount Roseby Schist, the Dugald River Shale Member, (host to the Ag-Pb-Zn deposit of the same name), and the overlying Lady Clayre dolomite (host to the Lady Clayre Cu-Au deposit), which has been dated at 1691±7Ma (Carson et al., 2011).

Sedimentation ended with the onset of the Isan Orogeny (approx. 1,600–1,510 Ma), which in its waning stages was accompanied by widespread emplacement of potassium-rich “A-type” granites. Williams and Naruku batholiths (approx. 1,550–1,500 Ma) are expo sed east of the Project area (Malakoff granite). IOCG mineralization has a close temporal relationship with granite formation, and it has been proposed that mineralizing fluids were generated though magma mixing and/or fractionation.

Sedimentation was reinitiated during the Cambrian, with deposition of fine- to medium-grained sandstones and limestone in basin grabens, including the Landsborough Graben located directly east of the Little Eva deposit.

7.1.2 Regional Deformation

Deformation of Proterozoic units within the Mary Kathleen domain resulted from the approximately 1,600–1,510 Ma Isan Orogeny. On a regional scale, the orogeny can be divided into three broad stages characterized by different principal stress directions and subsequent deformation responses. The Early Isan Orogeny (D1, approximately 1,600–1,570 Ma) was accompanied by north-south to northwest-southeast compression, which led to the formation of east-west trending folds and related axial plane cleavages. The Middle Isan Orogeny (D2, approximately 1,570–1,525 Ma) involved east-west compression, resulting in the development of north-south striking folds and foliation, ubiquitous in the Mary Kathleen domain. The Late Isan Orogeny (D3, approximately 1,525–1,500 Ma) represented a transition to dominantly brittle deformation, with the development of wrench-style faulting.

7.2 Project Geology

The Project area straddles the northern part of a north-south striking corridor up to 10 km wide and 80 km long, bounded to the east by the regionally significant Rose Bee Fault, and to the west by the Coolullah Fault, which is also the eastern bounding fault of the Phanerozoic Landsborough Graben. These faults terminate into the regional scale Fountain Range and Quamby faults, which continue south to intersect the Mary Kathleen domain's eastern margin (Figure 7-3 and Figure 7-4).

The Project area predominantly consists of variably metamorphosed sedimentary and igneous rocks of Proterozoic age that typically outcrop with limited residual regolith cover. Regolith cover tends to thicken east of the Rose Bee Fault and a thick sequence of Phanerozoic sediments overlies Proterozoic rock to the west of the Coolullah Fault in the Landsborough Graben. The graben contains Cambrian limestone and sandstone, mostly covered by Mesozoic and Cainozoic sediments.

Amphibolite facies schists of the Boomarra Metamorphic Belt are the oldest rocks within the area, outcrop east of the Rose Bee Fault (Figure 7-3), and are unconformably overlain by metamorphosed fine-grained sediments and intercalated volcanic rocks of the Corella Formation. The Little Eva copper-gold deposit is hosted by intermediate to mafic composition volcanic rocks within the Corella Formation, similar to rocks situated further to the south-east that have been dated, as coeval to the Wonga Suite intrusions (approximately 1,740 Ma).

The Knapdale Quartzite is a metamorphosed sequence of massive siliciclastics forming a prominent, 12-km long hill on the western side of the Project area, referred to as the Knapdale Range. The range is interpreted as a nappe structure, with east-directed thrust faulting juxtaposing older siliciclastics over younger Mount Roseby Schist (Roseby Schist).

The Roseby Schist, consisting of fine-grained, grey muscovite-quartz-biotite ± scapolite schists interbedded with carbonate-rich layers, has been structurally juxtaposed against the Corella Formation by major faults. The Roseby Schist within the Project area contains distinctive scapolite porphyroblast-rich units and is also distinguished by a lack of Wonga Suite felsic intrusions.

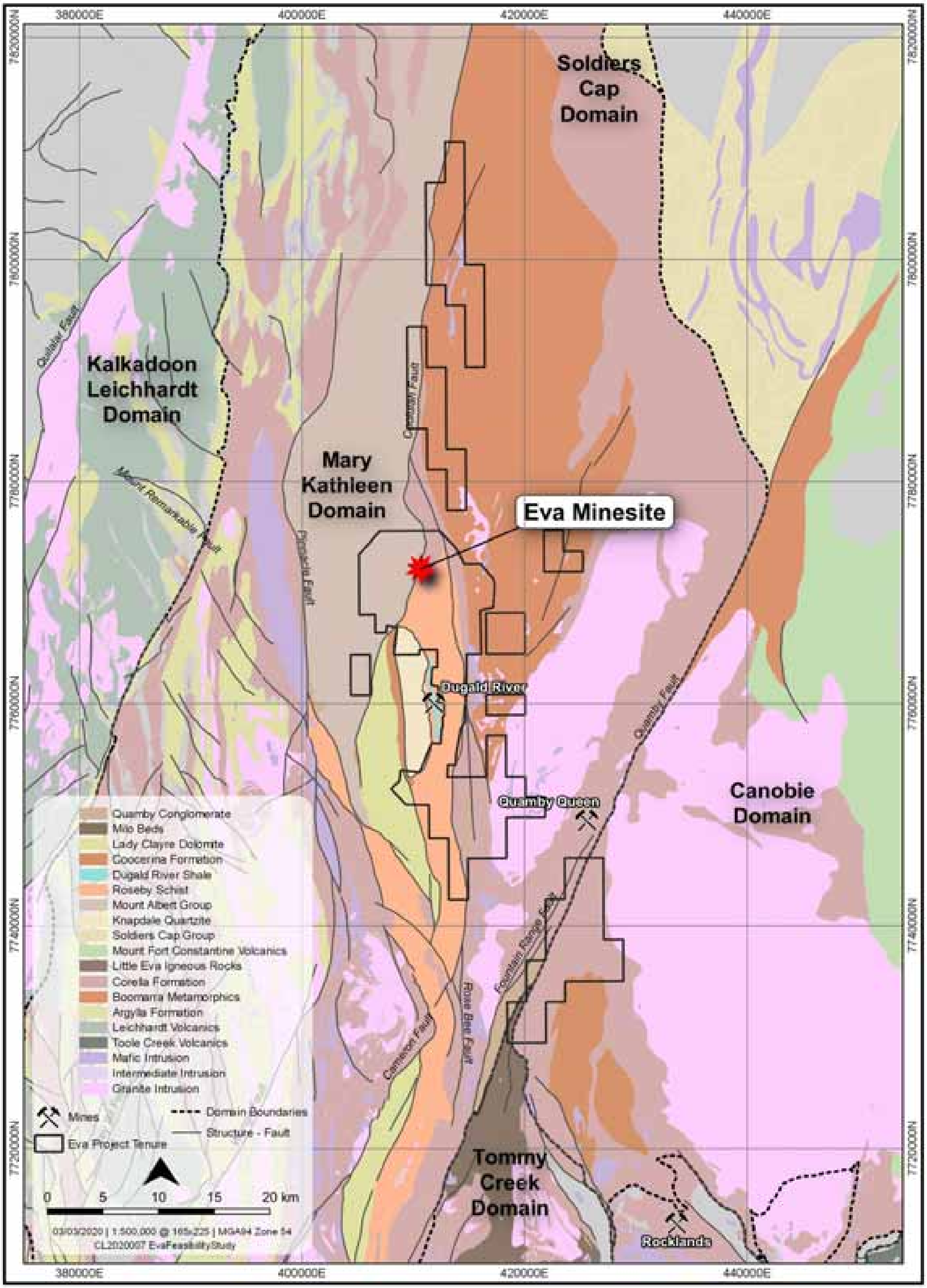


Figure 7-3: Geological Domains and Principal Stratigraphic Units of the Eva Copper Project Area

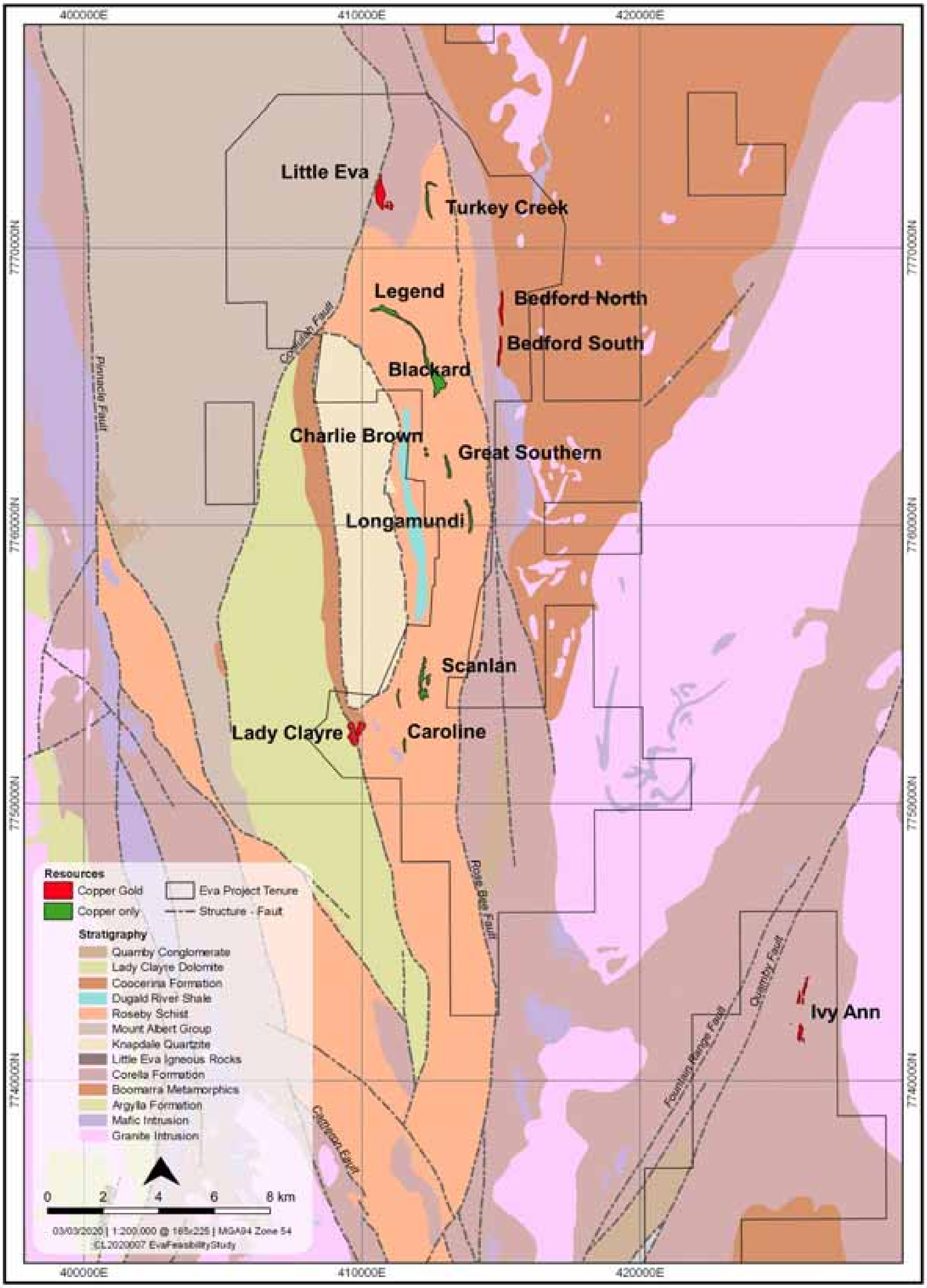


Figure 7-4: Project Area Geology with Outline of Project Tenure and Major Deposits

Overlying the Roseby Schist is the Dugald River shale member (carbonaceous zinc-rich slates), which hosts the Dugald River zinc-lead-silver deposit of 63 Mt at 12.5% Zn, 1.9% Pb, and 31 g/t Ag. The Dugald River deposit is localized along the highly deformed and faulted eastern margin of the Knapdale Range. On the western side of the Knapdale Range, similar zinc-rich shales occur in the Coocerina Formation, which is also overlain by dolomites, and therefore is likely a structural repetition of the Dugald River deposit host stratigraphy.

Dating indicates maximum ages for the Roseby Schist and the Dugald River Shale Member at approximately 1,686 Ma. The units have temporal equivalents (1,690–1,645 Ma) throughout the Mount Isa Inlier, which are host to many of the region's significant deposits, including Mount Isa, Hilton, Cannington, Lady Annie, Lady Loretta, Osborne, and Mount Elliot.

The Neoproterozoic (approx. 1,500 Ma) Quamby Conglomerate forms a ridge in the southern part of the Project area. Comprising polymictic conglomerate and medium- to coarse-grained sandstone, the Quamby rocks are relatively undeformed, generally flat-lying with broad open folds. The conglomerate unconformably overlies Corella Formation rocks in a small graben developed along the Rose Bee Fault during late Isan Orogeny wrench-fault reactivation. The conglomerate hosts gold mineralization that was initially mined by prospectors in the 1920s, and then later in the 1990s.

The Rose Bee Fault is a prominent topographic feature forming linear ridges where it is pervasively silicified and quartz-veined. Locally, the silicification overprints copper mineralization and may have developed during the Phanerozoic reactivation of the fault.

7.2.1 Little Eva Deposit Geology

The Little Eva deposit is currently the major example of hydrothermal IOCG mineralization and is the largest single copper deposit within the Eva Copper Project area. Little Eva is a close analogue of the Ernest Henry deposit. Measured and Indicated resources are 122 Mt grading 0.36% Cu and 0.07 g/t Au at a 0.17% Cu cut-off grade. Gold is strongly correlated with copper and is recovered in the copper concentrate. The deposit is 1.4 km in length and between 20 m to 370 m wide with mineralization extending from surface to the limits of drilling at 350 m vertical depth below surface (165 mRL) (Figure 7-5 and Figure 7-6). The deposit is sub-cropping on a flat plain with thin and variable (<3 m) in-situ soil and alluvium cover. Fresh rock is overlain by a 5 m to 25 m thickness of weathered rock. Copper occurs as primary sulphide minerals in fresh rock, and as secondary oxide minerals within the weathered zone.

Mineralization is hosted by a large body of faulted subvolcanic porphyritic and amygdaloidal intermediate rock that displays pervasive sodium and potassium feldspar, hematite, and magnetite metasomatic alteration assemblages. Intermediate volcanic rocks on the western margin of the deposit are cut by felsic intrusions that are also mineralized. Most of the mineralization is structurally controlled within breccias, fracture fill and veinlet stockworks. Chalcopyrite is the dominant copper mineral with lesser amounts of bornite. Mineralization is coarse and readily recovered through flotation concentration.

The igneous rocks hosting the Little Eva deposit occur within intercalated folded calc-silicate, marble, quartzite, and biotite-scapolite schists. The feldspar-phyric and amygdaloidal intermediate rocks are presumed to be volcanic flows, but probably include some subvolcanic sills as documented at Ernest Henry. In the northern part of the deposit the volcanic rocks are interpreted to be striking north and dipping to the east at approximately 60 to 70 degrees, while the mineralization appears to have a moderately west-dipping (45 to 65 degree) ladder-like grade distribution. In the central part of the deposit the volcanic stratigraphy is sub-vertical to westerly dipping, with dips shallowing to the south (Figure 7-6) The intrusive rocks are dominantly mafic to intermediate in composition, fine- to medium-grained with feldspar phyric and amygdaloidal textures. There is a minor porphyritic felsic intrusion along the western margin of the deposit. In plan, the intrusive rock package has a lenticular shape, imbricated by mineralized breccias and post-mineral faulting, and is enclosed by metasedimentary rocks. The western contact between igneous rocks and metasediments is, in part, highly strained and fractured. Copper mineralization is rare within the metasediments.

Folding and extensive cross-faulting have resulted in a complex array of fracturing, crackle brecciation, and veining, particularly within the more competent rocks associated with copper-gold mineralization. Late, post-mineralization, strong shearing, and fracturing occurs parallel to the footwall contact against calc-silicate rocks, and it is interpreted that more strain took place in the less competent rocks.

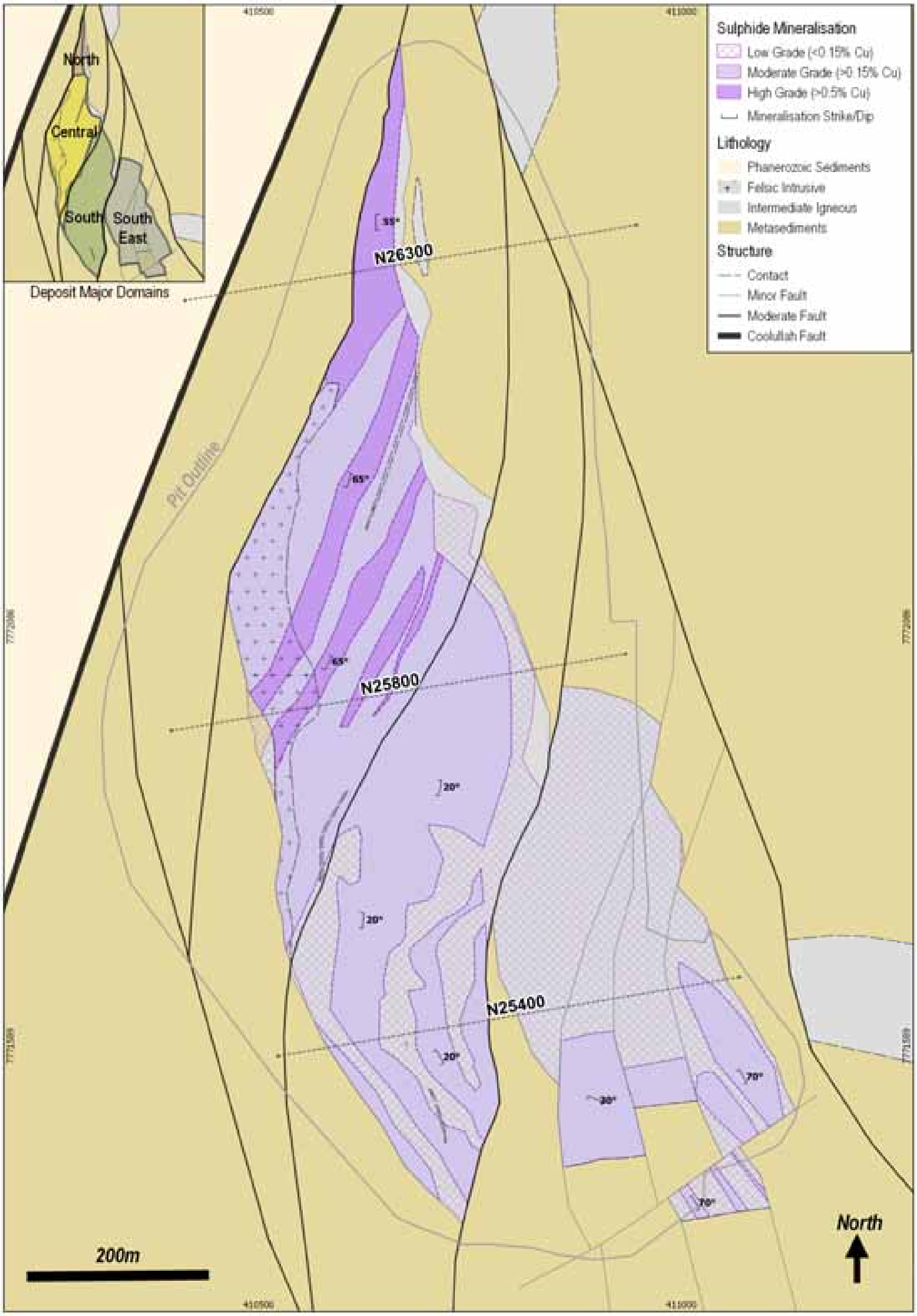
Copper-gold mineralization is high-grade but relatively narrow in the north and has progressively moderating grades associated with greater width in the southern half of the deposit. Higher-grade zones in the north occur in stacked zones of breccia, veining, and fracturing. Intervening zones are lower grade with disseminated and veinlet-hosted mineralization (Figure 7-7). The breccia zones typically dip west at 45 to 65 degrees, with north-northeast strikes. The breccias occasionally display multiple re-brecciation. Lower-grade mineralization in the south is more evenly distributed in fractures, veinlets, and disseminations. Low-grade mineralization averages 0.1% Cu to 0.3% Cu over lengths of 25 m to 150 m, whereas breccia zones are in the order 0.8% Cu and 0.12 g/t Au over widths of 15 m and display gradational contacts.

The mineralized intermediate rock is variably and pervasively altered by multiple stages of alteration. Initial alteration assemblages of amphibole, magnetite, and biotite (dark grey coloured) are overprinted by assemblages comprising albite, hematite, magnetite, and carbonate ± chalcopyrite (red coloured).

The mineralization is open beyond the extents of drilling: the northern tapered high-grade zone is terminated or offset by faulting or plunges steeply to the north; while the southern extent is poorly constrained by drilling, with higher-grade mineralization appearing to plunge to the south.

The sulphide mineralization is generally coarsely crystalline, and metallurgical tests have demonstrated recoveries greater than 95% for copper. No deleterious elements were present in the trial flotation concentrates. The deposit is generally low in sulphur and concentrations of pyrite greater than chalcopyrite are relatively rare. Many of the drill holes average less than 0.8% S.

A shallow 15-m to 25-m-thick oxidation profile has resulted from weathering and contains goethite-hematite with minor malachite, chrysocolla, covellite, azurite, neotocite, and cuprite. The weathering profile indicates a “dry” oxidation, as there is no leached zone and no supergene zone. There is a thin transition zone where predominately oxide copper changes over to predominately sulphide copper over 1 m to 2 m. Chalcopyrite can occur locally at surface. Zones of strong shearing and fracturing locally exhibit deeper oxidation.



Note: See Figure 7-6 for locations

Figure 7-5: Geology and Mineralization at the Little Eva Deposit

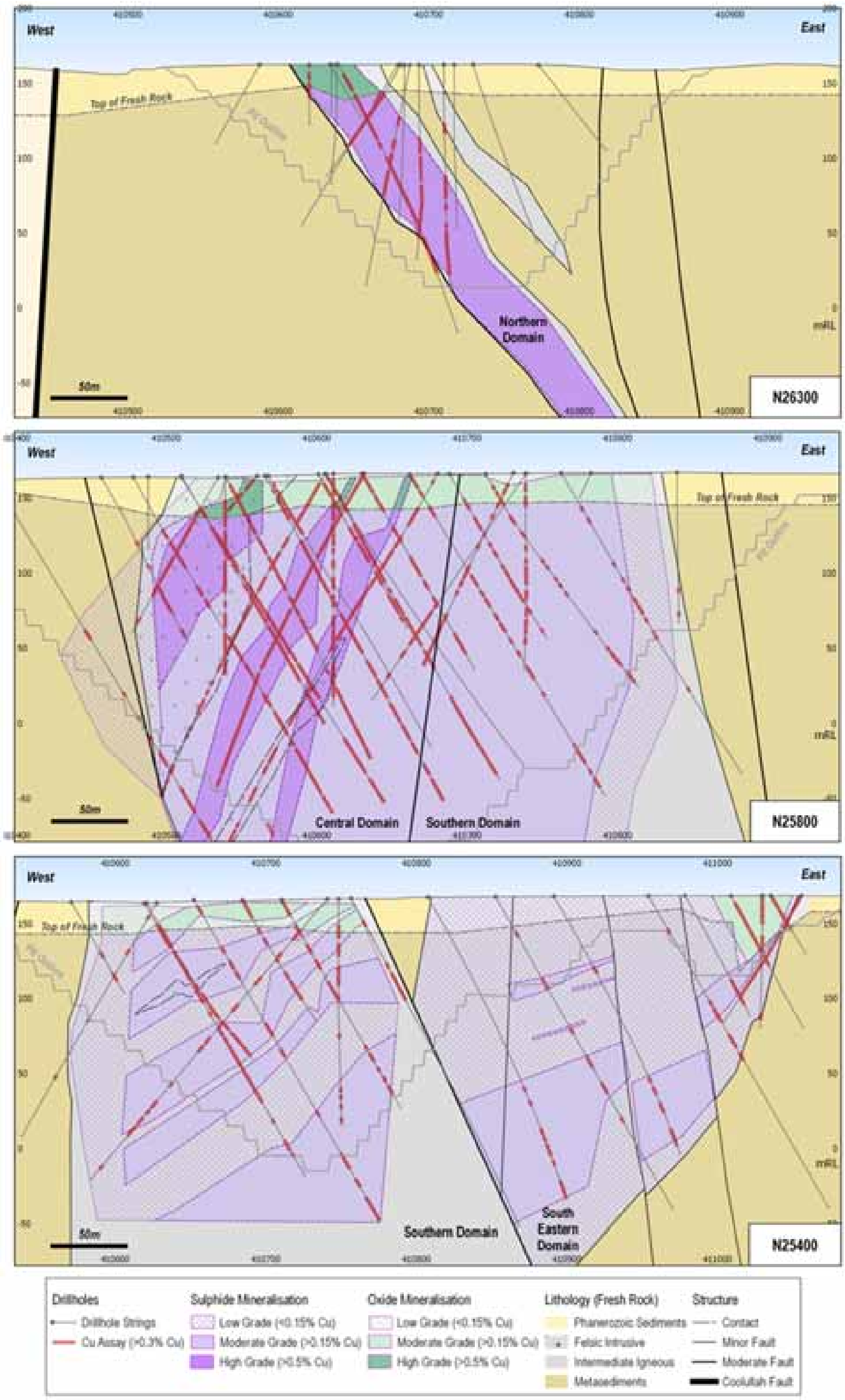


Figure 7-6: Geological Cross-Sections through the Little Eva Deposit from North to South

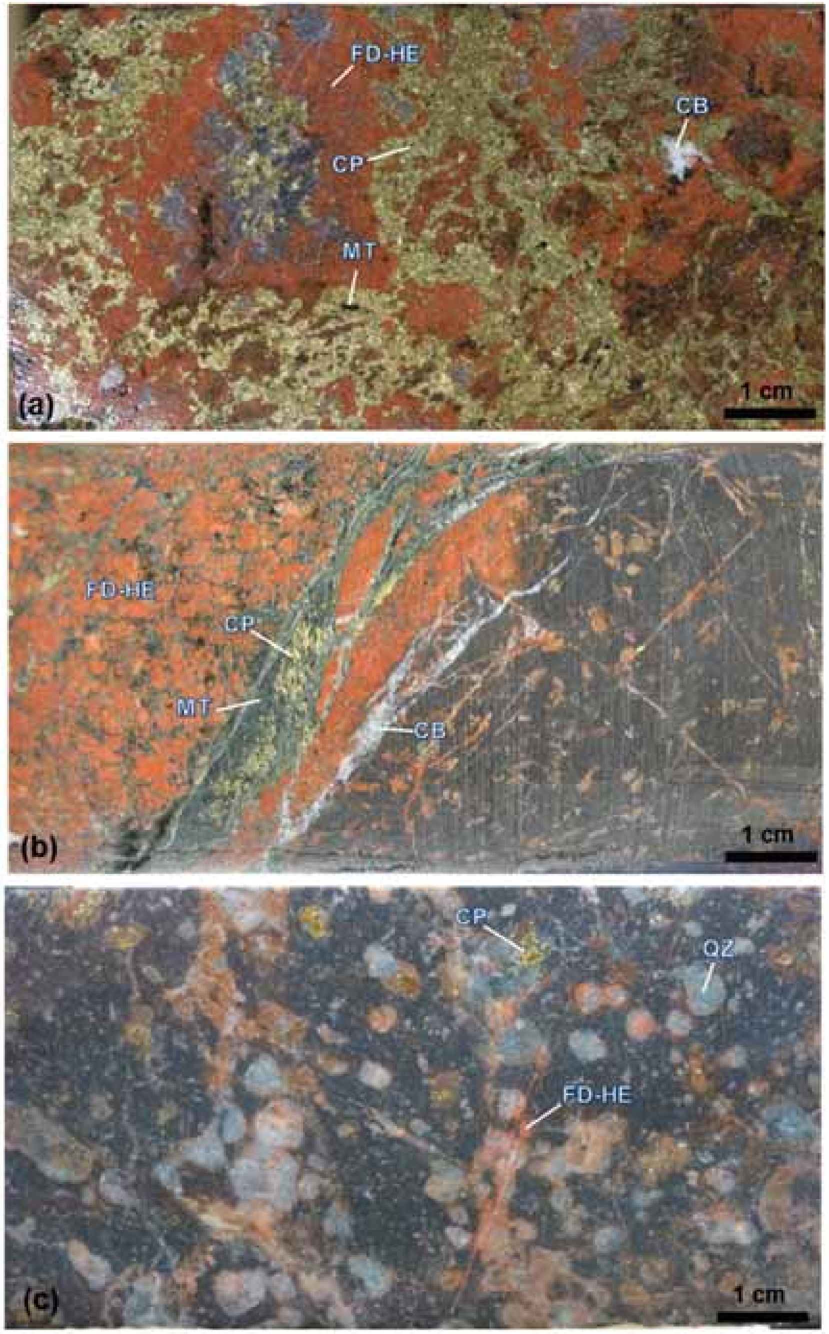


Figure 7-7: Drill Core Illustrating the Principal Mineralization and Alteration Styles at Little Eva Deposit

(a) High-grade hydrothermal breccia, with variably altered intermediate igneous clasts in a feldspar (FD), hematite (HE), chalcopyrite (CP), magnetite (MT), and carbonate (CB) matrix (4.8% Cu, 0.2 ppm Au).

(b) Feldspar phyric intermediate igneous rock (dark domain, right), overprinted by texturally destructive feldspar-hematite (FD-HE) alteration (red domain, left) host to a chalcopyrite (CP), magnetite (MT), and carbonate (CB) filled veinlet network (0.5% Cu, 0.05 ppm Au).

(c) Feldspar-phyric intermediate igneous rock with quartz (QZ) filled amygdales, patchy weak feldspar-hematite (FD-HE) alteration, and low-grade disseminated chalcopyrite (CP) mineralization (0.2% Cu, 0.02 ppm Au).

7.2.2 Turkey Creek

The Turkey Creek deposit (TCd) is the closest satellite deposit to the ore processing area that contributes to the mine plan. It is located 1.5 km east of the Little Eva deposit. The sulphide resources in the Measured and Indicated categories are 16.9 Mt grading 0.45% Cu with an additional 12.9 Mt grading 0.40% Cu in the Inferred category. Mineralization at Turkey Creek is very low in gold. The deposit is sub-cropping in a relatively flat to gently undulating area with thin (<0.5 m) in-situ soils and alluvium cover. Fresh rock is overlain by a 25-m to 90-m thickness of weathering and oxide mineralization. Copper occurs as primary sulphides in fresh rock and as secondary oxide minerals within the weathered zone.

The deposit extends over 1.8 km in length with mineralization extending from surface, to drilled depths of 150 m vertically below surface (Figure 7-8 and Figure 7-9), with a simple tabular geometry that displays excellent continuity along strike and down-dip. True widths vary from 10 m to 30 m at the southern end, to 30 m to 50 m at the northern end. The main part of the deposit strikes north and dips 60 degrees to the east. At the northern end, the mineralization and host stratigraphy are folded sharply eastwards into a curved synform shape which dips steeply south. The northern zone is slightly offset by faulting from the main southern zone.

The tabular deposit has an upper and lower zone of stronger copper mineralization with a more sporadically mineralized central zone. Primary copper mineralization comprises finely disseminated chalcocite, with subordinate bornite and chalcopyrite, that are disseminated and also occur within minor carbonate veinlets. Copper sulphide minerals in the upper zone are dominated by chalcopyrite, and in the lower zone by chalcocite and bornite. Gangue minerals primarily consist of quartz, calcite, scapolite, white mica, and minor biotite.

The sulphide mineralization is stratabound and hosted within a sequence of interbedded metasediments (biotite schists, biotite scapolite schists, and carbonate-rich rocks or marble) The host rocks are variably altered to carbonate and albite-hematite assemblages.

A consistent 20-m to 30-m thickness of weathering with oxide mineralization blankets the southern zone. It includes a zone of complete oxidation, and a thin transition zone with minor secondary and remnant primary copper sulphides. Copper oxide mineralization comprises minor malachite, rare occurrences of azurite, and native copper, with most of the native copper thought to be associated with hydrobiotite similar to the Blackard deposit. The transition zone is dominated by malachite, minor degraded chalcopyrite, chalcocite, and rare native copper.

Weathering is deeper in the northern zone, extending to 70 m to 90 m deep. Copper minerals are dominated by copper silicates (chrysocolla, hydrobiotite) and minor malachite.

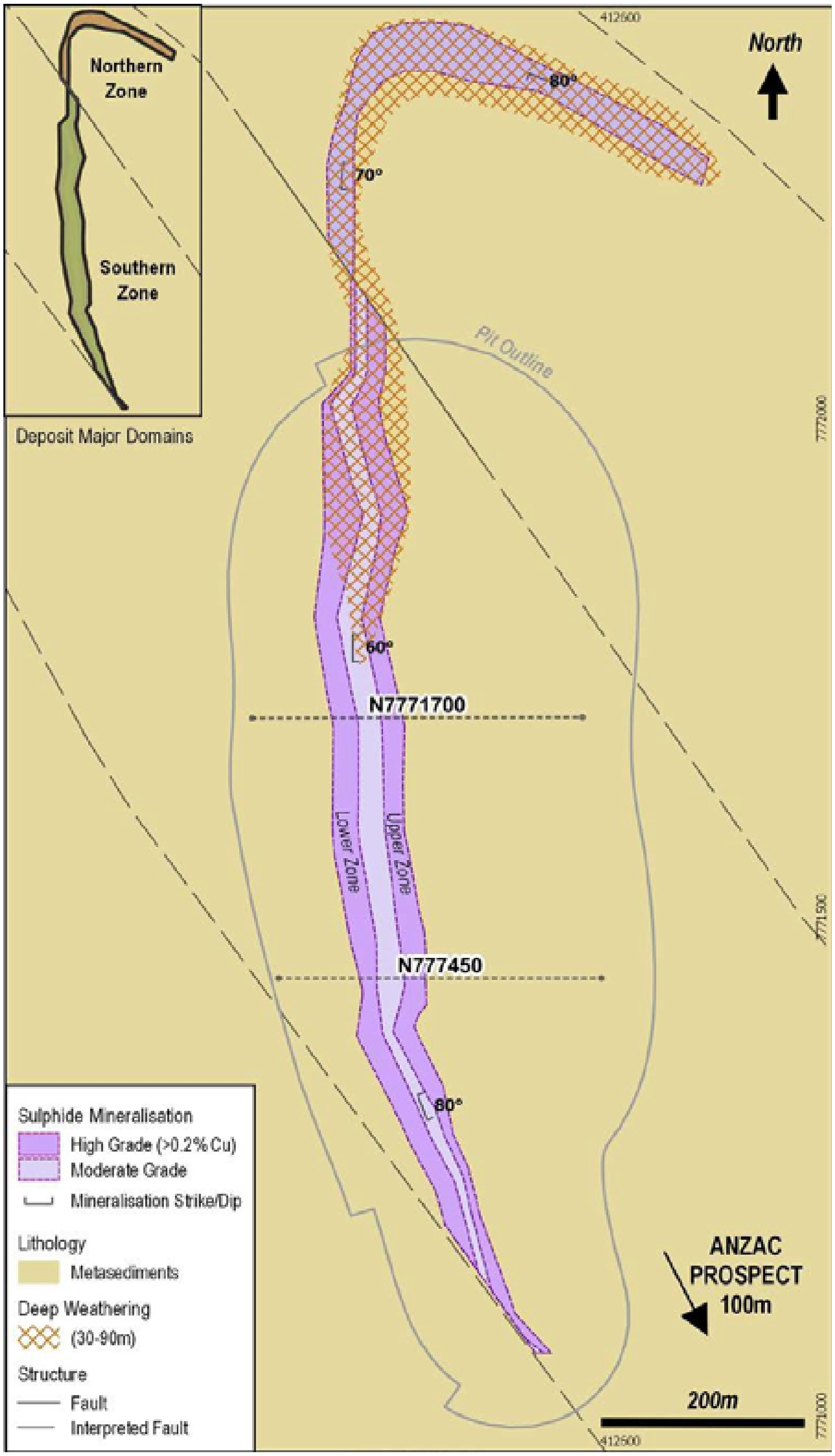


Figure 7-8: Turkey Creek Deposit Mineralization

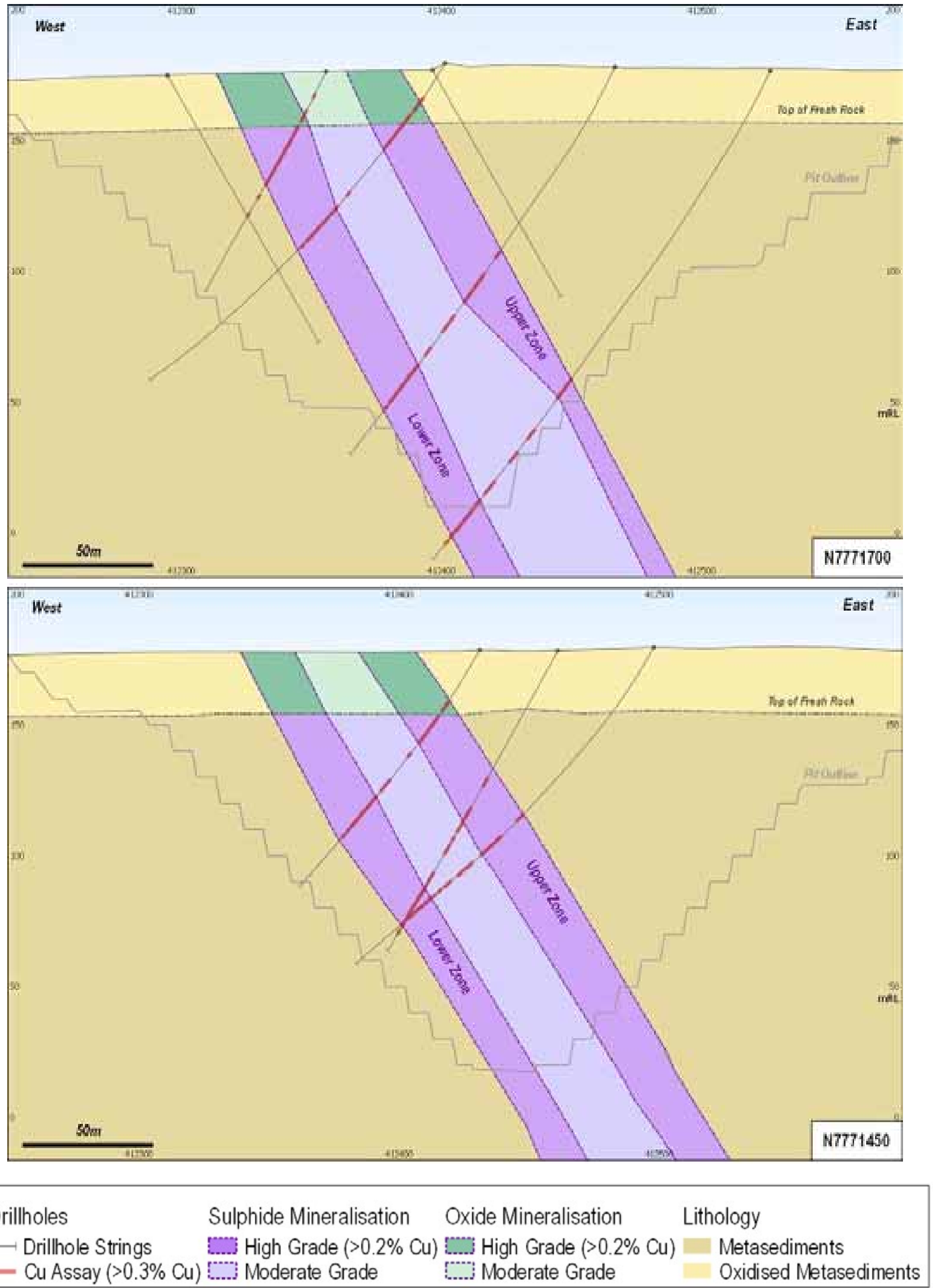


Figure 7-9: Geological Cross-Sections through the Southern Zone of the Turkey Creek Deposit

7.2.3 Blackard and Scanlan

The Blackard and Scanlan deposits are located approximately 5 km and 17 km, respectively, south of the Eva deposit. The deposits are geologically very similar and therefore are described together. A thin northerly extension of mineralization from the Blackard deposit is called the Legend deposit, which is sparsely drilled and containing only Inferred resources.

An additional 18 RC drill holes were completed on the Blackard deposit in 2019, and extensive metallurgical testing was carried out on Blackard and Scanlan samples which has defined metallurgical recoveries for the mineralogical zones within the deposits, thereby allowing reserves to be defined. Measured and Indicated resources for the Blackard deposit are 76 Mt grading 0.49% Cu using cut-off grades of 0.24%, 0.20% and 0.17% for the copper, transitional, and sulphide zones, respectively. Measured and Indicated resources for the Scanlan deposit total 22 Mt grading 0.57% Cu using cut-off grades of 0.26, 0.20 and 0.17% for zones as listed above.

The Blackard and Scanlan deposits are hosted by the Mount Roseby Schist, a map unit which contains intercalated marls and carbonaceous sediments, representing a shallow marine to lagoonal depositional environment that has been metamorphosed to calc-silicates, and variable scapolite, biotite and/or muscovite schists. The host rocks have undergone polyphase deformation with the most significant folding event forming northerly-trending folds, likely coinciding with peak amphibolite grade metamorphism. Fold geometry has been inferred from data collected from diamond drill core and field mapping, and has been variably described as isoclinal, through tight to open, depending upon location, but primary layering cannot be determined from RC chips and is only rarely visible in drill core, making interpretation at the deposit scale difficult. The Scanlan through to Blackard-Legend deposits form a 7 km long trend of mineralization that appears to follow stratigraphy as it curves around the east side of the Knapdale Quartzite (Figure 7-4).

The Blackard deposit morphology is a function of folded stratigraphy and/or faulting having a strike length of 3.5 km and a maximum plan width of 350 m (Figure 7-12). The stratigraphic width of the deposit is only 60 m to 90 m, but a series of parasitic folds and/or fault repetitions results in a much wider deposit. Fault movement along axial planes may have resulted in rootless folds. The southern area of the deposit is relatively narrow, steeply dipping to the west, and northerly trending. The deposit width and depth extent increases to the north, with a gradual shallowing of the westerly-dipping mineralization (45 degrees) and a flattening of mineralization to the east. It is, however, difficult to constrain the mineralized rock within a symmetrical fold pattern and the slight variations in strike orientation of higher-grade zones in plan suggest the possibility of an east-west stacking of mineralization along possible north-south (~010^{o} N) faults. To the north, the deposit narrows to a moderately-dipping 50 m to 60 m thick band that gradually steepens and thins northwards.

The Scanlan deposit has a strike length of 1,500 m and a maximum width in plan of 500 m (Figure 7-13). In the southern half the deposit is composed of a 10 m to 50 m thick horizon, with the thicker part folded into a “V” shaped synform on the eastern side, and the thinner part forming a nearly flat antiform to the east, resembling an extended square root symbol in section, with a 320 degrees northwesterly trend. The east dipping part of the synform is not present in the northern part of the deposit, eroded or possibly faulted off, and the mineralization swings to a 12 degrees northerly trend, becoming a steeply west-, then east-dipping panel of mineralization, and gradually thinning to uneconomic widths.

Mining and processing of Blackard and Scanlan deposits will be affected by the deep weathering profiles, which has resulted in extensive modification of the host rock and localized remobilization of copper. Much of the carbonate has been leached from the upper parts of the deposits creating voids between less soluble or insoluble mineral grains and reducing the mass of the rock. Copper released by oxidation of sulphide minerals has mostly formed native copper particles many of which are very fine-grained. Some of the copper occurs as ultra-fine particles (<10 µm) within altered biotite (termed hydrobiotite) which is unrecoverable with any known commercial processing methods. Four zones defined by weathering and copper speciation have been determined for the deposits, and extensive testing has determined probable metallurgical recoveries for each zone. From upper to lower, the zones are:

- Oxide Zone. The deposits are capped by a weathered, ferruginous zone that is typically 20 m to 30 m thick and has a sharp contact with the next underlying zone. In some areas of the Oxide Zone almost all copper has been leached but other areas have significant copper grades, with copper occurring as malachite, azurite, hydrobiotite, and Fe-Mn-Cu mineraloids known as neotocite. Testing suggests copper in this zone it is not economically extractable.
- Copper Zone. The Copper Zone is defined by the presence of native copper with lesser cuprite, copper-bearing hydrobiotite, and chalcocite. Leaching of carbonates has reduced the mass and created a very soft rock. The Copper Zone has a variable thickness, reaching a maximum of 120 m. Extensive testing has defined a viable process for extracting a significant percentage of the native and sulphide copper.
- Transition Zone. A relatively narrow zone ranging from 1 m to 15 m in thickness that marks the transition from the Copper Zone to the Copper Sulphide Zone and carries mineral phases of both adjacent zones. Copper grades tend to be high due to the presence of supergene chalcocite. The base of this zone is defined by the “top of fresh rock” (TOFR in Figure 7-11).
- Sulphide Zone. Defined by unweathered (fresh) rock with copper sulphide species of bornite, chalcocite, chalcopyrite, and pyrite, this zone contains sulphide disseminations and clots which are strongly associated with carbonate veinlets. Metallurgical recoveries from this zone are favourable. Silver is locally present but was not estimated.

There are two possibilities for the origin of mineralization in the copper-only deposits. The first ascribes a hypogene hydrothermal source that occurred during the waning stages of the Isan Orogeny due to features that include orientation of sulphide minerals along foliation planes and/or brittle fractures or pre-existing carbonate veins, as well as sulphide-phased overprinting metamorphic minerals. Timing of this mineralizing event would closely correspond with the copper-gold deposits in the district. The second hypothesis for the mineralization is that the deposits represent typical stratiform copper deposits that form from metalliferous basin brines, post-deposition but pre-orogeny. Stratiform-type copper deposits are typically formed by redox reactions within marine sediments with moderate to high sulphur contents. These deposits commonly display an inwards pyrite-chalcopyrite-bornite-chalcocite-native copper zonation as the redox reactions progressively use up the available sulphur; a zonation that may be inferred based on the deeper, down-dip parts of the Blackard deposit. Additionally, the lower sulphide zones within the copper-only deposits have virtually no gold but relatively high silver contents, locally, with Cu:Ag ratios typical of many occurrences of stratiform copper mineralization. A stratiform origin may also explain the similar stratigraphic position of all the copper-only deposits (with the exception of Turkey Creek) around the Dugald River Shale and Knapdale Quartzite units. Many of the sulphide textures ascribed to the hypogene origin are compatible with metamorphism of earlier formed sulphide deposits within carbonaceous rocks, where sulphides and some carbonate would be partially remobilized and likely to recrystalize after formation of the metamorphic silicate minerals. The extensive leaching of carbonate from the upper parts of these deposits indicates the possibility of weathering of an overlying high-sulphide zone (pyrite-chalcopyrite) to produce the necessary acid. The origin of the deposits is inconsequential to grades and mining but may have some significance for future exploration.

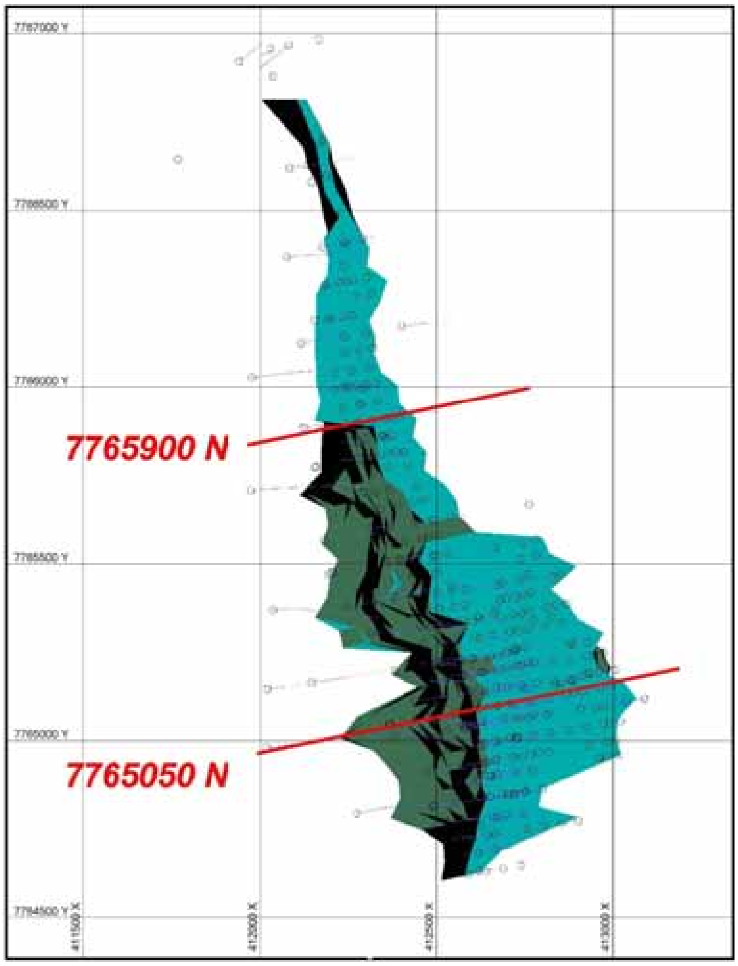
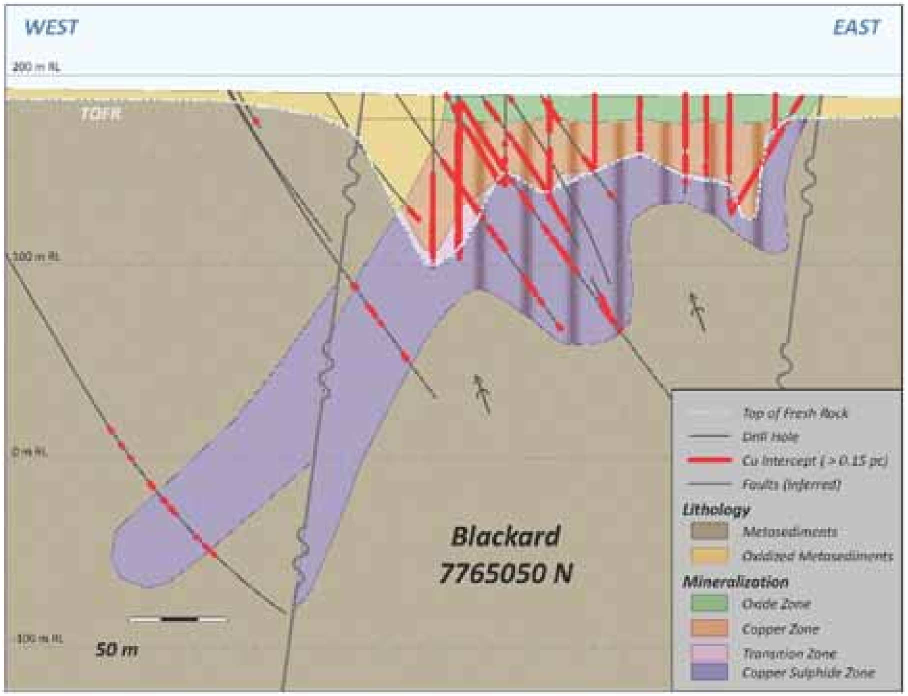


Figure 7-10: Plan View of the Blackard Deposit with Location of Cross-Sections



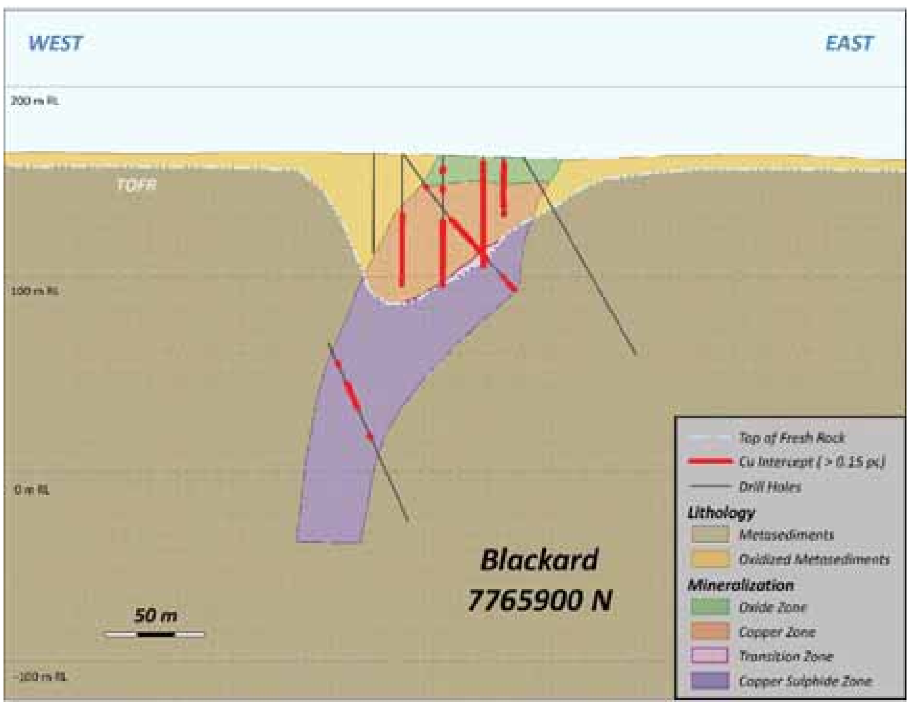


Figure 7-11: Geological Cross-Sections through the Blackard Deposit Illustrating the Distribution of Mineralogical/Metallurgical Zones Produced by Weathering

Note: The upper photograph is an example of core from Oxide Zone containing ferruginous metasediments with clay alteration associated with leaching of carbonate. The lower photograph is core from the copper zone consisting of chemically oxidized scapolitic schist, leached of carbonate. Copper assay values for 1 m samples shown. Horizontal field of view approximately 70 cm.

Figure 7-12: Photographs of Drill Core from the Blackard Deposit

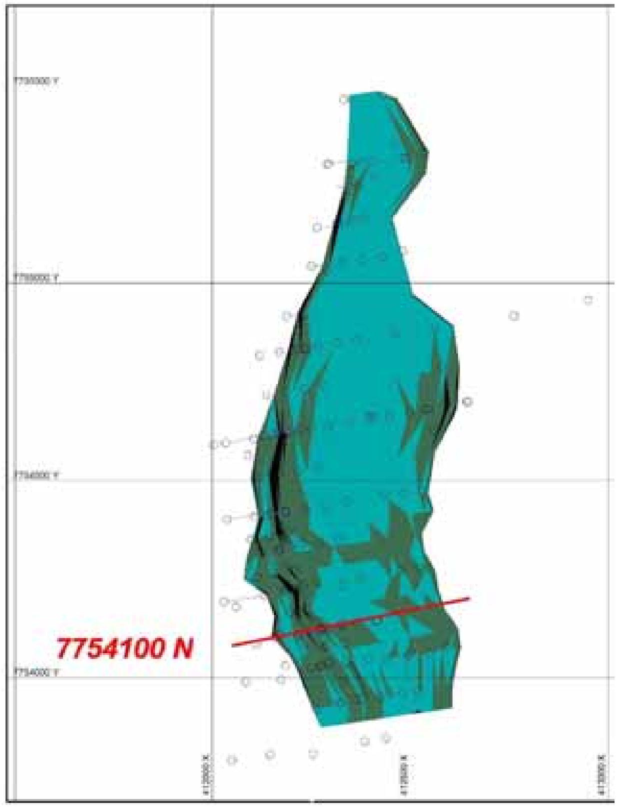


Figure 7-13: Plan View of the Scanlan Deposit with Cross-Section Line

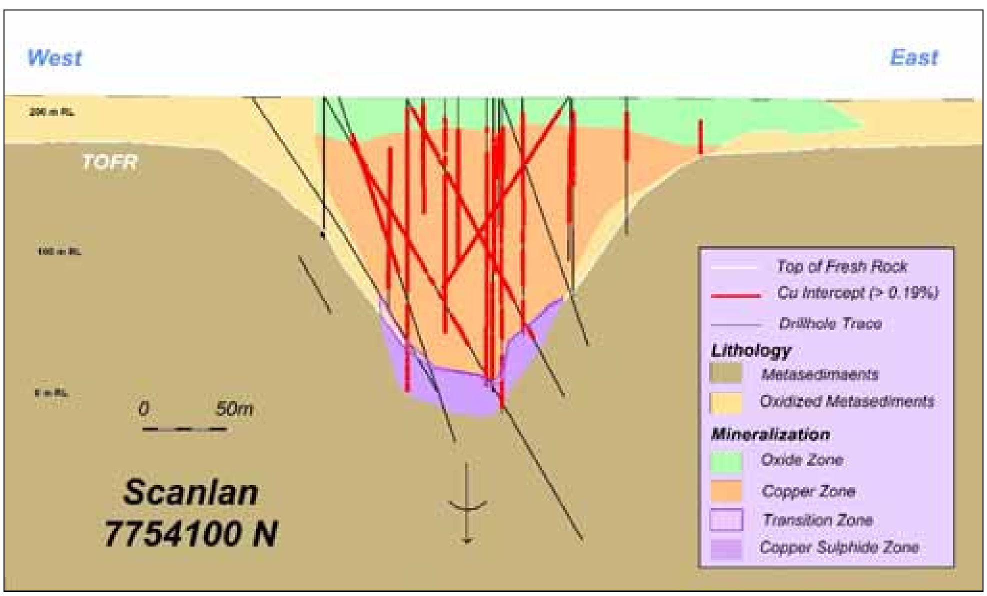


Figure 7-14: Cross-Section of Scanlan Deposit Illustrating Mineralization Zones

7.2.4 Lady Clayre

Lady Clayre is the third largest copper-gold deposit within the Project area. It is located approximately 19 km south of Little Eva. The deposit contains Measured and Indicated resources of 7.3 Mt grading 0.41% Cu and 0.17 g/t Au, plus an additional 5.0 Mt grading 0.36% Cu and 0.0.15 g/t Au in the Inferred category. The deposit has been drilled to a vertical depth of 200 m and is open at depth. Lady Clayre is located close to the junction of two regional faults near the southern termination of the Knapdale Quartzite.

The deposit is sub-cropping in an undulating area with largely thin (<0.5 m) in-situ soils. Fresh rock is overlain by a thin, 15 m to 25 m, weathered zone of oxide mineralization. Copper occurs as primary sulphides in fresh rock and as secondary oxide minerals within the weathered zone.

Mapping and surface sampling have defined multiple zones of surface mineralization. Zones A and F (Figure 7-15 and Figure 7-16) have been the focus of drilling, which has delineated a series of moderate to steep dipping planar mineralized bodies. Zone A mineralization strikes north-northwest, dips approximately 80 degrees to the west, and extends along strike for 700 m. Zone F mineralization strikes north-east, dips 70 to 75 degrees to the west, and extends along strike for a total of 480 m.

Lady Clayre is situated in a structurally complex area, with evidence for a number of ductile and brittle deformation events. Copper-gold mineralization is structurally controlled, associated with faulting/shearing in Zone F sub-parallel to bedding in a folded sequence of shale, metasiltstone, schist, and dolomite. The metasedimentary package is intruded by intensely altered, narrow (0.5 m to 5 m) sheets of mafic intrusive. Alteration mineral assemblages associated with mineralization are dominated by carbonate, feldspar, quartz, and tremolite.

The main sulphide ore mineral is chalcopyrite, often associated with lesser pyrite and/or pyrrhotite. Molybdenite is also noted. Mineralization is coarse-grained, occurring in sulphide or carbonate-sulphide vein arrays and as sulphide disseminations in intensely altered rocks. Breccia infill can also be locally significant.

An irregular, 15 m to 25 m thick zone of weathering with oxide mineralization blankets the deposit. The dominant copper oxide mineral is malachite, with limonite and goethite.

Mineralization remains open along strike and down dip in Zones A and F, while a series of additional areas of surface mineralization remain untested by drilling

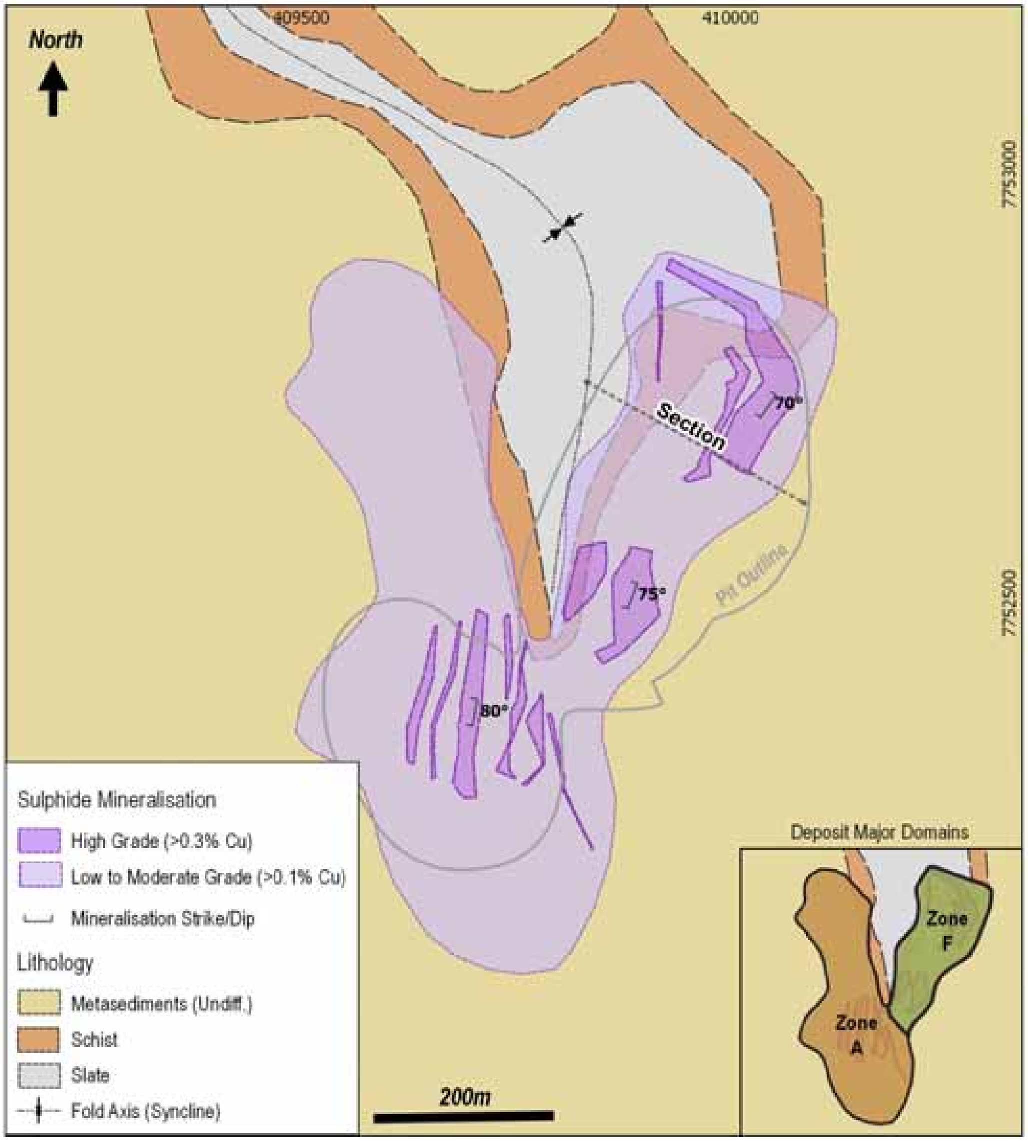


Figure 7-15: Geology and Mineralization at Lady Clayre

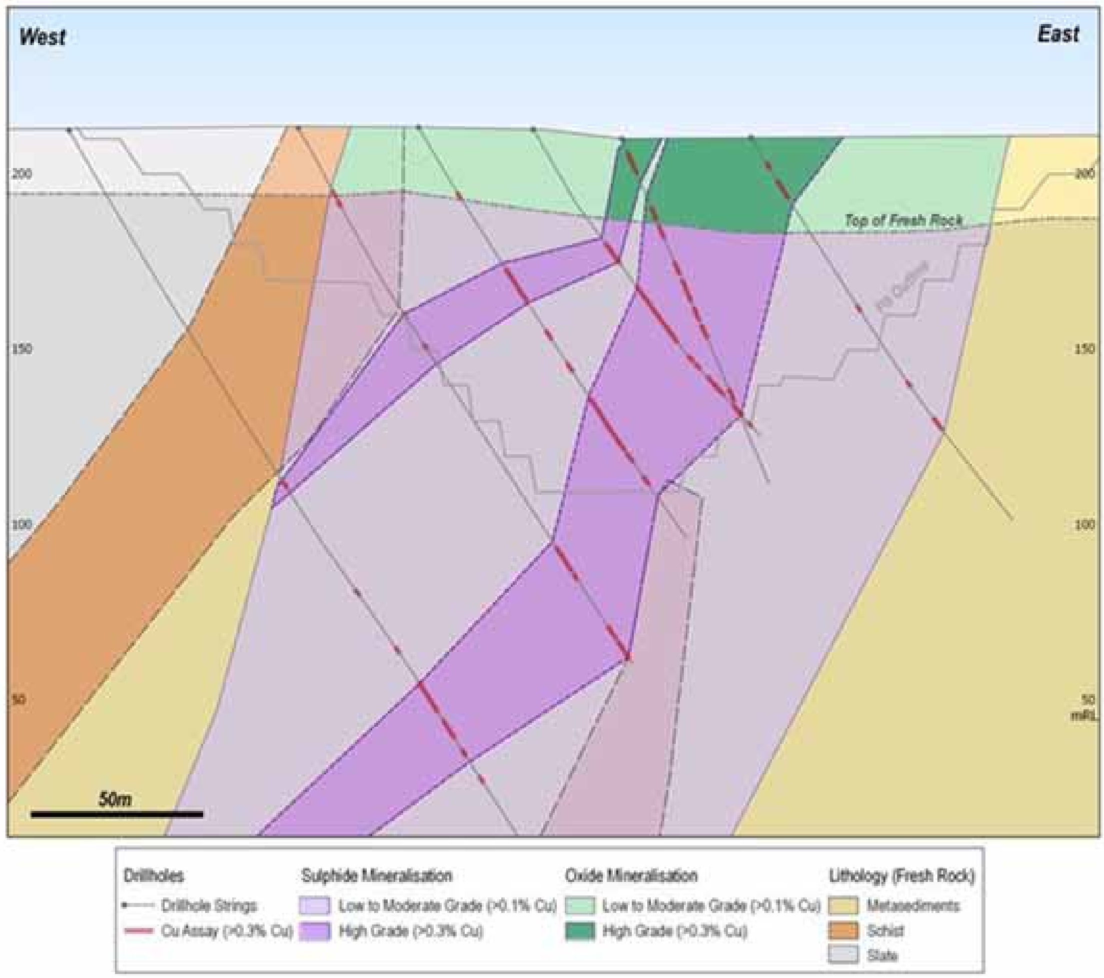


Figure 7-16: Geological Cross-Section through the Lady Clayre Deposit Zone F

7.2.5 Ivy Ann

Ivy Ann is a modest sized copper-gold deposit located approximately 35 km south-southeast of Little Eva. The Measured and Indicated Resource is estimated at 5.1 Mt at 0.36% Cu and 0.08 g/t Au at a 0.17% Cu cut-off grade. The deposit has been drilled to a vertical depth of 125 m and is open at depth. Ivy Ann lies to the east of, and adjacent to, the broad Quamby Fault Zone, which is manifest as a 1-km wide high-strain zone with evidence for dextral displacement (Figure 7-17).

The deposit is sub-cropping in a relatively flat to gently undulating area with largely thin (<0.5 m) in-situ soils and transported alluvium cover. Fresh rock is overlain by a thin 15-m to 30-m-thick weathered zone of oxide mineralization. Copper occurs as primary sulphides in fresh rock and as secondary oxide minerals within the weathered zone.

The deposit is a lenticular shaped body striking north-northeast with numerous lenses hosted within steeply east-dipping structures, striking north-south to north-north eastly. Mineralization has been defined in two separate deposits, Ivy Ann and Ivy Ann Norths. The overall mineralization extends over a strike length of 3 km. The main Ivy Ann deposit is defined over a strike length of 630 m, with a width of 20 to 130 m, and a steep easterly dip; it's a wedge-shaped body striking north-south subparallel to the host lithologies. The Ivy Ann North deposit is defined over a 420-m strike length with a width of 10 to 30 m and is vertical or dips steeply to the east.

The copper-gold mineralization is fault hosted and associated with breccias and networks of veins and micro-veinlets within a folded sequence of metamorphosed sediments (psammite) and amphibolite. Fold axes are north-south with interpreted moderate southward plunges (>45 degrees). Main sulphide ore minerals are chalcopyrite with lesser pyrite and pyrrhotite. Sulphide grain size is relatively coarse.

Alteration mineral assemblages associated with the copper mineralization are dominated by albite, quartz, hematite, biotite, and magnetite. Breccias are best developed in albite-quartz-hematite altered rocks, which sit in the hinge of a tight southward-plunging antiform. The metasediments, fault zones, and fold axes are cut by a swarm of thin (<5 m) pegmatite dykes.

An irregular 15-m to 30-m thick zone of weathering with oxide mineralization blankets the deposit. The dominant copper oxide mineral is malachite, present with goethite and hematite, and lesser amounts of chrysocolla, tenorite, and cuprite. The zone is poorly constrained by current drilling.

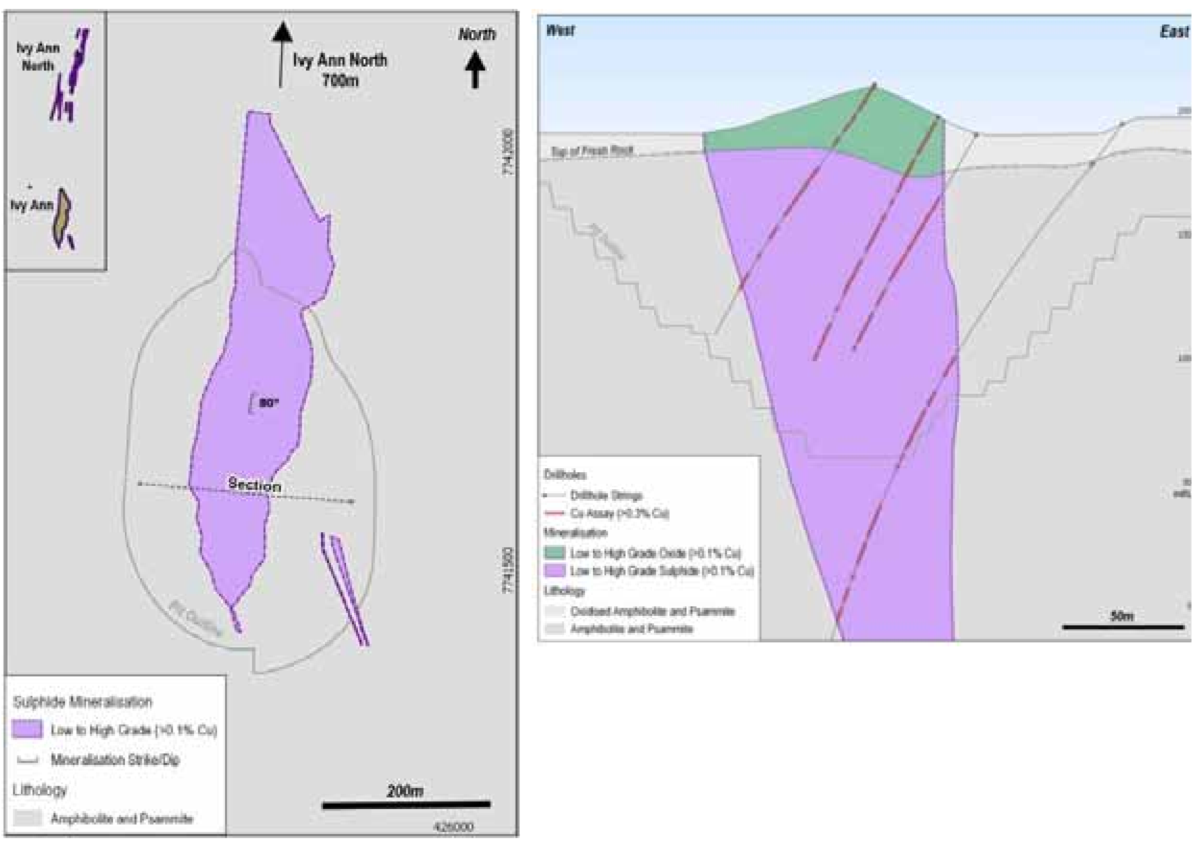


Figure 7-17: Plan of Ivy Ann Mineralization and Geological Cross-Section of the Ivy Ann Deposit

7.2.6 Bedford

Bedford is a modest sized copper-gold deposit located 6 km southeast of the Little Eva deposit. The Measured and Indicated Resource is estimated at 3 Mt at 0.54% Cu and 0.14 g/t Au at a 0.17% Cu cut-off grade. The deposit has been drilled to a vertical depth of 140 m and is open at depth. Bedford lies to the east of the Rose Bee Fault.

The deposit is sub-cropping in a relatively flat to gently undulating area with thin (<0.5 m) soils and limited alluvium cover. Fresh rock is overlain by a 20-m to 30-m-thick weathered zone of oxide mineralization. Copper occurs as primary sulphides in fresh rock, and as secondary oxide minerals within the weathered zone.

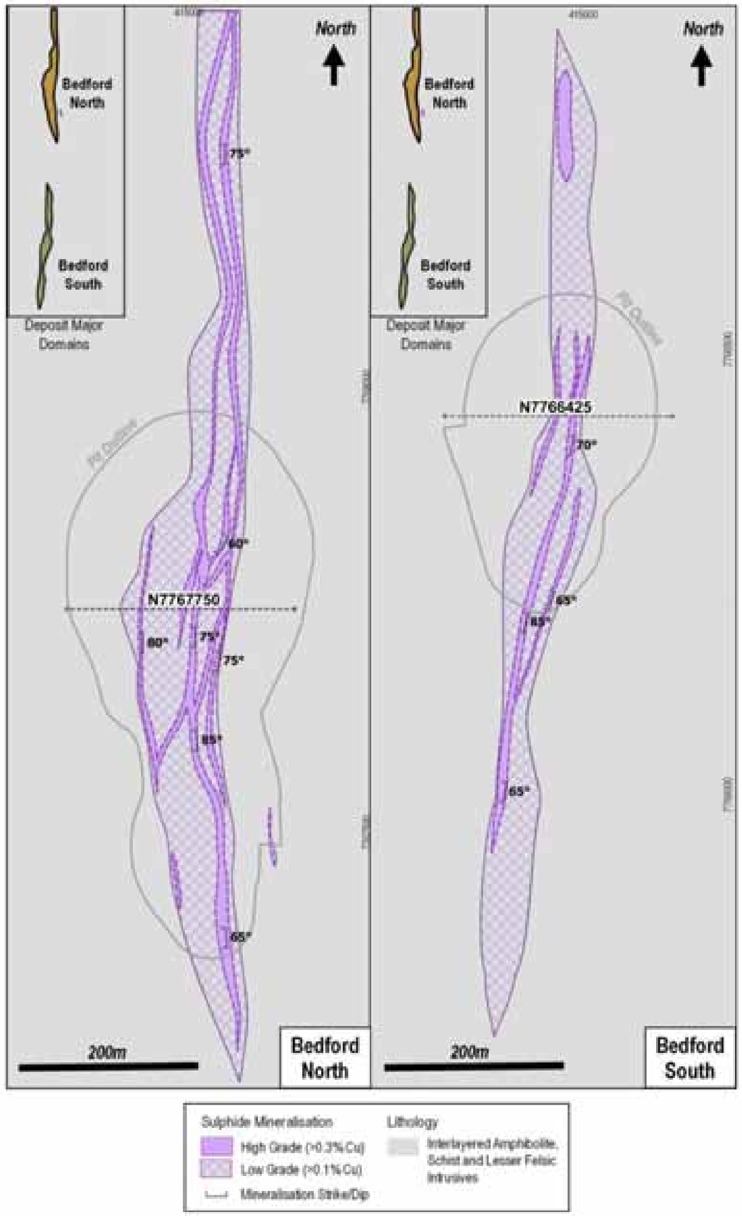
The deposit is hosted within a steeply west-dipping shear zone striking north to north-northeast (Figure 7-18 and Figure 7-19). The shear zone varies from 50 m to 120 m wide with internal arrays of mineralized structures and splays. Mineralization has been defined in two separate zones, Bedford North, and Bedford South, within a continuous structure. The deposit extends over a strike length of 2.5 km. The northern zone is 1.15 km, and the southern zone is 850 m long. Within the shear zone individual mineralized structures associated with ore grade mineralization (>0.3% Cu) have true widths ranging from 5 m to 12 m.

Host rocks are a north to north-northeast-striking, moderately to steeply west-dipping interlayered sequence of amphibolite and biotite schist underlain by psammite and intruded concordantly by narrow planar granite and pegmatite dykes or sills. In Bedford South, mineralized structures are interpreted to be bedding or foliation parallel. In Bedford North, the main mineralized structures are interpreted to trend north-south, stepping across the north-northeast-striking stratigraphy, with the development of a set of secondary north-northeast linking structures along bedding or foliation. Magnetite-biotite alteration assemblages with quartz veining are concentrated in the ore zones, above a strongly feldspar-hematite altered footwall.

The dominant ore mineral is coarse-grained chalcopyrite (with minor magnetite, pyrite, pyrrhotite, and gold), which occurs within quartz veins, breccia fill, and disseminations within the host shear zone.

An irregular, 20-m to 30-m-thick zone of weathering with oxide mineralization blankets the deposit. Although the base of oxidation is well defined, variability of copper mineral species within the weathering profile is not well understood.

Mineralization remains open to north and south along strike, down dip, and between the two zones.



Note: In Bedford North the main mineralized structures trend north-south stepping across north-northeast-striking stratigraphy of intercalated amphibolite, biotite schist, and narrow granite and pegmatitic dykes/sills. In Bedford South the mineralized structure is bedding or foliation parallel.

Figure 7-18: Bedford Deposit Mineralization Plan

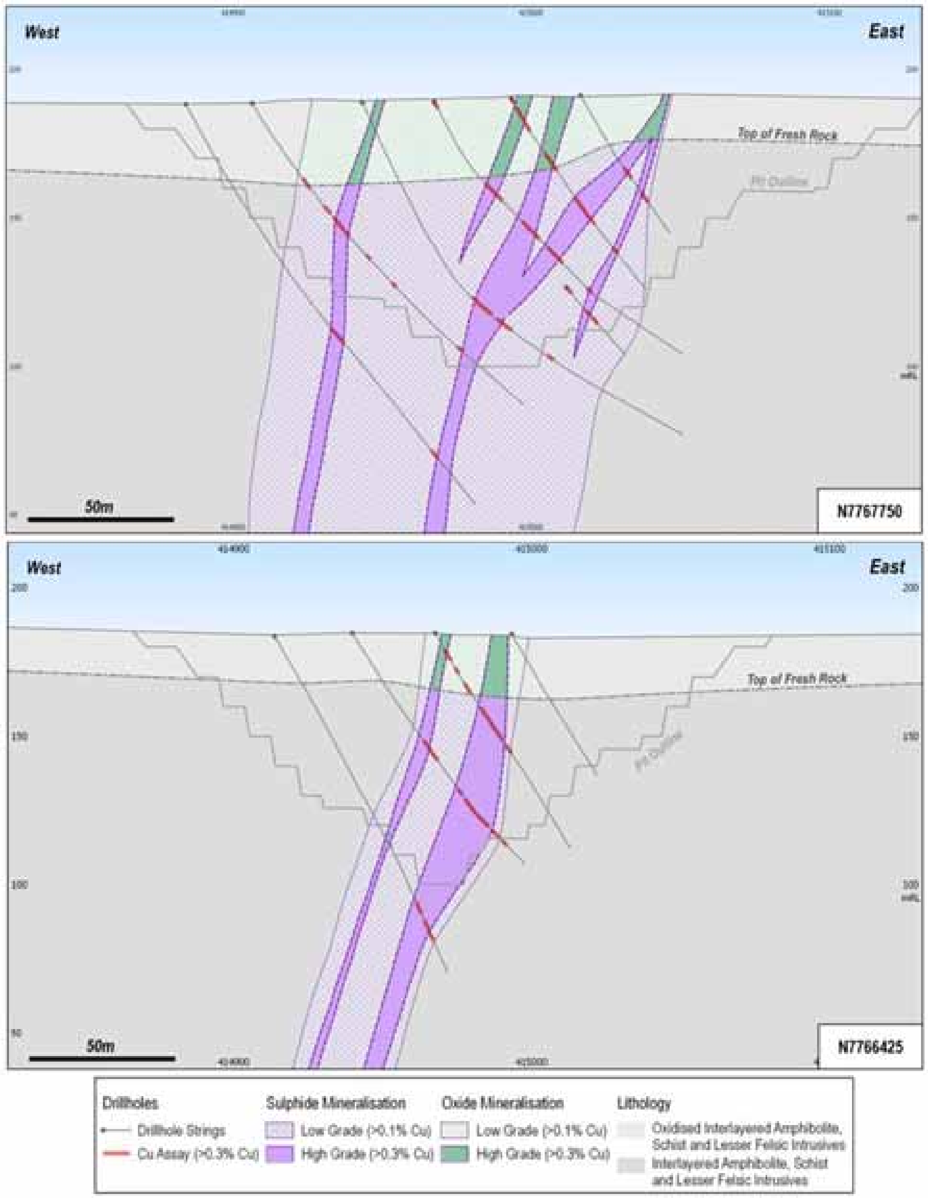


Figure 7-19: Geological Cross-Sections through the Bedford Deposit

8 DEPOSIT TYPES

The copper-gold deposits within the Project are of the IOCG style of hydrothermal mineralization. Significant examples of Australian IOCG deposits include Olympic Dam and Prominent Hill in South Australia and Ernest Henry in Queensland, which is located some 60 km from Little Eva.

Mineral deposits occurring within IOCG systems are associated with relatively high temperature, iron-rich hydrothermal alteration (typically hematite or magnetite), which is both spatially and temporally related to felsic plutons. Mineralization can manifest in a variety of styles including vein networks, breccias, disseminations, and replacements. Deposits are typically localized in dilation zones of structures active during pluton emplacement and cooling.

Within the Eastern Mount Isa Inlier, deposits are interpreted to have formed during the waning stages of the Isan Orogeny (1,530–1,495 Ma), in association with intrusion of the Williams-Naraku batholith suites. This is coincident with wrench reactivation of earlier large, crustal-scale faults, which saw dextral displacement on north-northwest trending transfer faults, and some regional north-south structures, suggesting northwest-southeast compression.

In the Project area, deposits fit into two categories: copper-gold, and copper-only. The copper-only deposits are a distinct, metasediment-hosted stratabound mineralization style in the region, unique to the Roseby Schist. The copper-gold deposits are more typical of the IOCG deposits in the Eastern Mount Isa Inlier. The copper-gold deposits occur within structural-lithological settings that facilitate dilational sites during deformation, typically within igneous rocks or intercalated metamorphosed igneous and sedimentary rocks peripheral to Roseby Schist. The copper-only deposits are interpreted from the available data to be gold-poor end members of the IOCG mineralizing event prevalent throughout the district (varying primarily due to host rock controls) an alternative hypothesis is that they are stratiform deposits related to an earlier mineralising event during basin dewatering.

8.1 Copper-Gold Deposits

Four copper-gold deposits are scheduled for mining: Little Eva, Lady Clayre, Ivy Ann, and Bedford (which contains two separate zones, Bedford North, and South).

Little Eva, the largest copper deposit within the Project, is considered an IOCG type, and is a close analogue of the Ernest Henry deposit. The deposit contains gold, which has a strong correlation with copper, and is recovered in the copper concentrate. The deposit is hosted by a large body of faulted, porphyritic, and amygdaloidal intermediate rock, which likely represents volcanic flows, and possibly sub-volcanic intrusive rocks. All rocks display pervasive sodium and potassium feldspar, hematite, and magnetite-bearing metasomatic alteration assemblages. The mineralization is structurally controlled within breccia and veinlet stockworks. Chalcopyrite is the dominant copper mineral. Mineralization is generally coarse-grained, and readily recovered through flotation concentration.

Bedford, Lady Clayre, and Ivy Ann have a similar metal association to Little Eva. These are smaller shear zone-, fault-, and vein-hosted deposits within thinly intercalated metasedimentary and igneous rocks. Gold grades within these deposits are typically higher than at Little Eva.

All the deposits are sub-cropping, covered by a relatively shallow (approximately 25 m) oxidized cap.

8.2 Copper-Only Deposits

There are three copper-only deposits which are planned for mining: Turkey Creek, Blackard, and Scanlan. Other copper-only type deposits within the Project tenures currently excluded from the mine plan, as they are currently insufficiently explored, are: Legend, Longamundi, Great Southern, Caroline, and Charlie Brown. The copper-only deposits contain trace amounts of gold locally, but generally not in economic quantities as with the copper-gold deposits. Low tenor silver may be present in the sulphide zones, although data is minimal. The mineralization appears to be stratabound, if not stratiform, and in the case of Blackard and Scanlan has been deformed by folding. Except for Turkey Creek, these deposits are distributed around the eastern margin of the Knapdale Range over a strike length of 16 km and hosted within a sequence of metamorphosed calcareous sediments. The deposits are not associated with magnetite enrichment and exhibit some characteristics of stratiform-copper type deposits. Primary sulphide mineralization is dominated by bornite, with minor chalcopyrite and chalcocite, however the deposits have been modified by supergene processes and extensive leaching of carbonate, that has produced four distinct mineralogical zones as listed below:

- The oxide zone begins at surface, and extends to depths of 15 m to 25 m. The zone is defined by oxidation of copper and iron bearing minerals to malachite, limonite, goethite, and copper bearing Fe-Mn mineraloids (neotocite).
- The copper zone occurs below the oxide zone, and can extend to depths of 100 m. The copper zone contains a significant amount of native or metallic copper, which can account for up to 65% of the contained copper. Significant copper also occurs in the lattice of altered biotite referred to as hydrobiotite, which is not recoverable by flotation. Other copper minerals include cuprite, chalcocite and residual bornite. Carbonate is extensively leached. Almost complete leaching of carbonate has produced very friable rock.
- The transition zone is a zone that transitions between the copper and sulphide zones. This zone contains minor secondary and remnant primary copper sulphides (chalcocite, cuprite, tenorite, bornite, and chalcopyrite), and may contain some metallic copper.
- The sulphide zone, is primary mineralisation in fresh rock containing copper as disseminated bornite, chalcocite and chalcopyrite.

The copper-only mineralization is associated with a specific stratigraphic interval that has ubiquitous low-tenor copper anomalism wherever it is exposed or intersected by drilling and displays complex folding and fault patterns. Fold axes are predominantly north-northwest-trending but can have variable plunges. At Blackard and Scanlan, mineralization occurs within shallow-plunging anticlines, with steeply-dipping to locally overturned western limbs, and flatter, east-dipping limbs.

9 EXPLORATION

Early exploration in the area that contributed significantly to the database for this Project included that completed by Ausminda Pty. Ltd., CRA Exploration (CRAE), and Pasminco, with later exploration by Altona, prior to Project acquisition by Copper Mountain.

Extensive geophysical surveying, primarily induced polarization (IP) over the copper deposit areas, and Electromagnetic (EM) or Controlled Source Audio-Frequency Magnetotellurics (CSAMT) over the Dugald River zinc deposit host rocks, as well as gravity and magnetic surveys, were undertaken in the area by CRAE. All the Project deposits subcrop and were initially identified by surface sampling and mapping. The most valuable result from the geophysical work was the identification aided definition of the copper-only deposits, the most valuable were the EM and gravity surveys. Gravity lows are registered over the copper-only deposits due to deep weathering, while the metallic copper in the supergene zones were mapped as EM anomalies. Airborne magnetic surveys over the Project area are available from various government agencies. Satellite hyperspectral surveys have also been used with some success by various companies in the area.

CRAE's bedrock and soil geochemical programs outside the Roseby copper deposits were not systematic, with minimal assessment of gold mineralization, and left most of the surrounding area untested by geochemical surveys. CRAE’s focus at the time was on the copper only (no gold containing) deposits due to their relatively high grades and the Little Eva and Lady Clayre areas were of secondary exploration interest. The Little Eva copper-gold prospect was drilled by CRAE to an Inferred resource status, but the gold content was not assessed. The Lady Clayre prospect was also drilled by CRAE at the time, but no resource estimate was completed. Metallurgical sampling and testing were conducted at Blackard and Lady Clayre, but not at Little Eva.

Following the acquisition of the Project from CRAE by Pasminco/Zinifex, drilling, and sampling programs focused primarily on the Lady Clayre copper-gold sulphide prospect, Caroline (Lady Clayre East), and the copper-gold potential of the Mount Rose Bee Fault area. This drilling was insufficient to define a formal resource at either deposit. Pasminco also initiated a soil and rock sampling program designed to examine the Mount Rose Bee Fault and related splay faults. While this program detected widespread but weak copper-gold mineralization, generally in close spatial relationship with copper and gold soil geochemical anomalies, Pasminco divested the Roseby Copper Project before the exploration program was completed.

Xstrata conducted exploration in the central Roseby area under the terms of an option and earn-in agreement with Altona. Xstrata also completed deep drilling below the Little Eva, Blackard, Great Southern, and Longamundi deposits demonstrating the presence of large mineralized systems. Xstrata also discovered a mineralized system under cover at Cabbage Tree Creek some 3 km north of Little Eva.

Xstrata has also completed extensive geochemical, rock sampling, mapping, and geophysical surveys generating numerous targets, some of which have been subject to initial drill testing with positive results.

Altona carried out systematic soil geochemistry work over much of the claim area, and this work is being continued by the Company. This work has established numerous copper-in-soils targets within the Project tenure and surrounding Exploration Permit for Minerals (EPM) held by CMMC (Figure 9-1). Shallow drilling of these by Altona, continued by the Company, has established numerous mineralized positions with opportunities to established new copper and gold mineral resources.

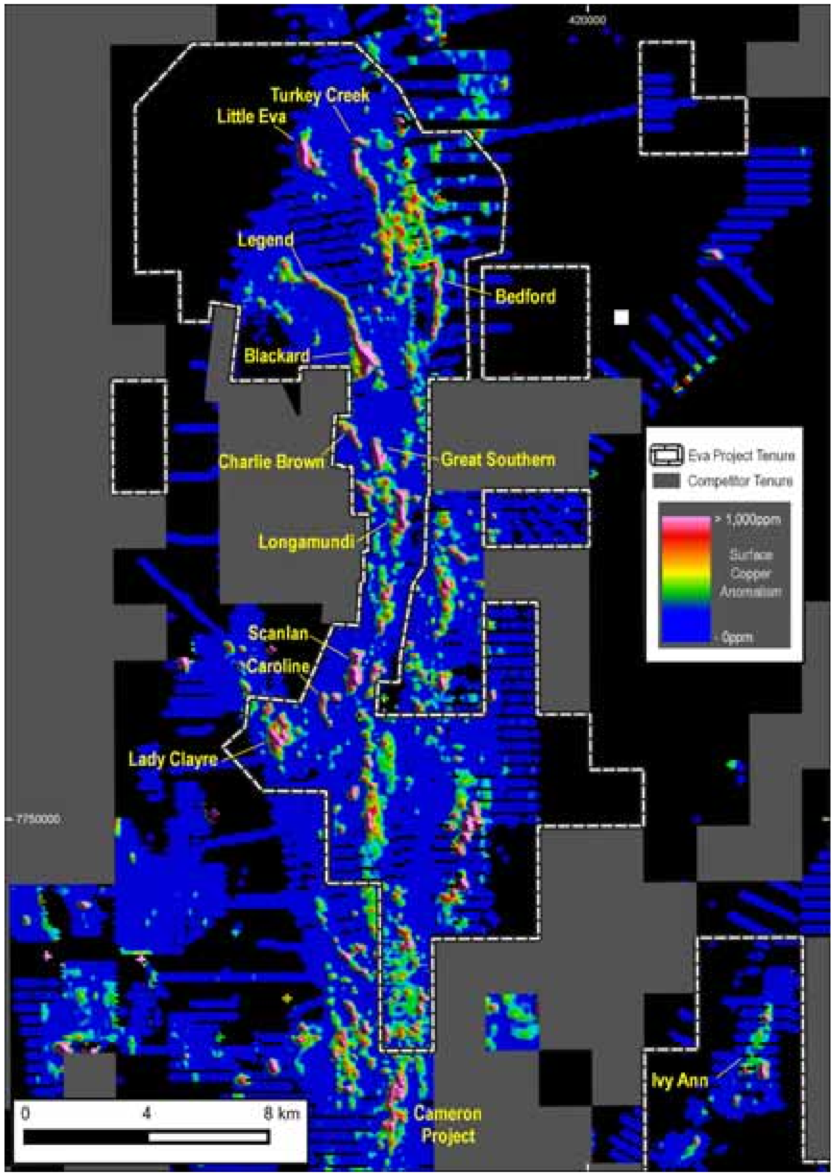


Figure 9-1: Surface Copper Anomalism with Defined Deposits and the Cameron Project Area Indicated

Exploration carried out by CMMPL in 2018 and 2019 included grade confirmation and metallurgical drilling in the Little Eva, Turkey Creek, and Blackard deposits, in addition to exploration drilling on some targets in the Project area. Additionally, exploration drilling was also completed on the prospective areas Quamby and Matchbox, which are located in the Company's Cameron area south of the Project area (Figure 9-1). Compilation of geophysical surveys and inversion of historical IP geophysical data were completed, as were new surveys in a few areas. Testing of aquifers for potential water sources near the proposed mine area was successfully conducted in both 2018 and 2019.

10 DRILLING

The seven deposits that are the focus of this study have relatively lengthy exploration histories, including multiple periods of drilling implemented and managed primarily by three companies: CRA Exploration (CRAE), Universal Resources Limited (Universal), and Altona. All drill data was collected to industry standards, and the procedures were well documented. Quality control and data verification are discussed in following sections, and demonstrate that the data is reliable and suitable for Mineral Resource and Mineral Reserve estimations. The information below was compiled from Altona's documents, and is included for completeness.

10.1 Drill Hole Data Description

10.1.1 Little Eva

A total of 77,226 m of drilling in 516 holes was completed at Little Eva. Of these, some 86% are reverse circulation (RC) (448 holes), and 14% are diamond drilling (75 holes). Holes were inclined at -55 to -60° or subvertical, generally drilled on 50 m spaced section lines, and 40 m along line spacing. Some areas are more densely drilled or include holes aligned in alternative directions. Diamond drilling was conducted for resource definition, metallurgical testwork sampling, geotechnical, and twinning of RC holes for quality assurance/quality control (QA/QC). Diamond drill holes were commonly drilled with shallow RC pre-collars.

The drilling history for the Little Eva deposit is summarized in Table 10-1, and hole locations are shown in Figure 10-1. The earliest recorded drilling at Little Eva was undertaken by CRAE in 1963, and consisted of a single diamond drill hole (DDH). Most drilling was conducted by three companies; CRAE (1963 to 1998), Universal (2002 to 2009), and Altona (predecessor to CMMPL) (2011 to 2018).

RC drilling typically utilized face sampling hammers (5.25", 5.5", or 6") and diamond drilling mainly provided NQ or HQ core samples. Where necessary, substandard data from early, poorly documented programs or drill methods with low, or poorly documented sample quality (e.g., costean, auger, or rotary air blast), or assay quality (e.g., partial or incomplete).

Table 10-1: Little Eva Drilling Summary

Year	Company	Hole Type	Hole Count	Metres Drilled
1963	CRAE	DD	1	193
1977	CRAE	DD	1	254
1978	CRAE	DD	5	1,159
1988	CRAE	RC	24	823
1990	CRAE	RC	5	480
1992	CRAE	DD	1	543
1992	CRAE	RC	12	1,182
1994	CRAE	RC	13	1627
1995	CRAE	DD	3	757
1995	CRAE	RC	6	1,031

Year	Company	Hole Type	Hole Count	Metres Drilled
1996	CRAE	DD	3	1,201
1996	CRAE	RC	1	150
2002	Universal	RC	14	2,138
2003	Universal	RC	5	1,249
2004	Universal	RC	83	9,987
2005	Universal	DD	18	2,698
2005	Universal	RC	147	20,875
2006	Universal	RC	34	3,633
2006	Universal	DD	12	1,338
2006	Xstrata	DD	2	984
2007	Universal	DD	10	1,103
2011	Altona	RC	104	21,085
2011	Altona	DD	7	2,041
2015	SRIG	DD	2	480
2015	Altona	DD	2	51
2018	CMMC	DD	1	164
Total			**516**	**77,226**

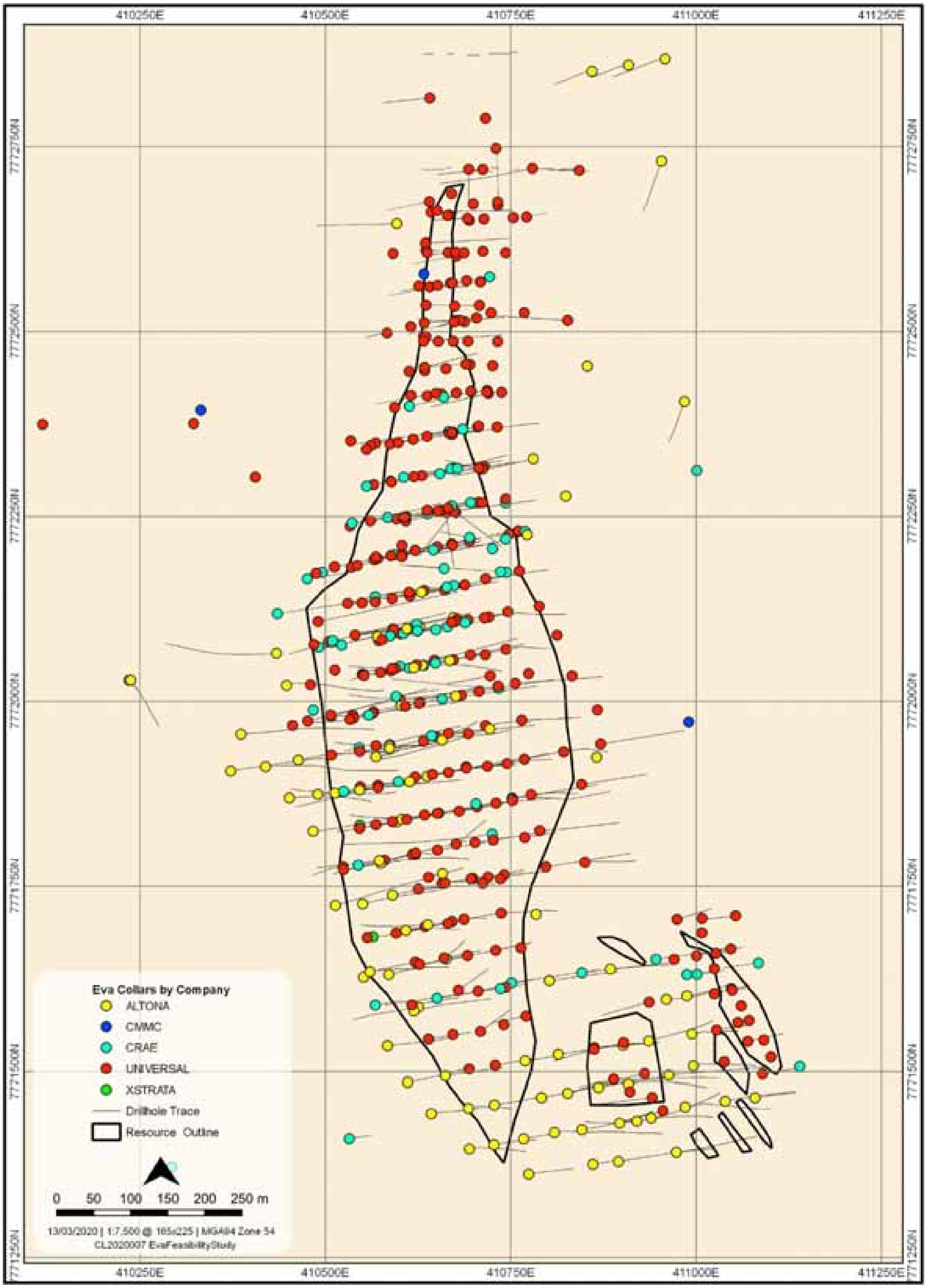


Figure 10-1: Little Eva Drill Collar Plan

10.1.2 Turkey Creek

A total of 8,218 m of drilling in 58 holes was completed at Turkey Creek. Of these, some 91% are RC (53 holes), and 9% are DDHs (5 holes). Holes were typically inclined at -60° and drilled along 100 m spaced section lines with 50 m spacing between drill holes. Diamond drilling was conducted for the primary purpose of metallurgical testwork sampling and geotechnical data.

The drilling history is summarized in Table 10-2, and hole locations are shown in Figure 10-1. The earliest hole at Turkey Creek area was a diamond hole drilled by Carpentaria Exploration in 1963, but the location details of this hole are uncertain, and the hole has been disregarded. The majority of drilling was conducted by Altona (now CMMPL) from 2012 to 2015.

RC drilling typically utilized face sampling hammers (5.5"), and diamond drilling provided either NQ or HQ core samples.

Table 10-2: Turkey Creek Drilling Summary

Year	Company	Hole Type	Hole Count	Metres Drilled
1993	CRAE	RC	2	218
2011	Xstrata	RC	2	300
2012	Altona	RC	7	1272
2014	Altona	RC	42	6,024
2015	Altona	DD	5	404
Total			**58**	**8,218**

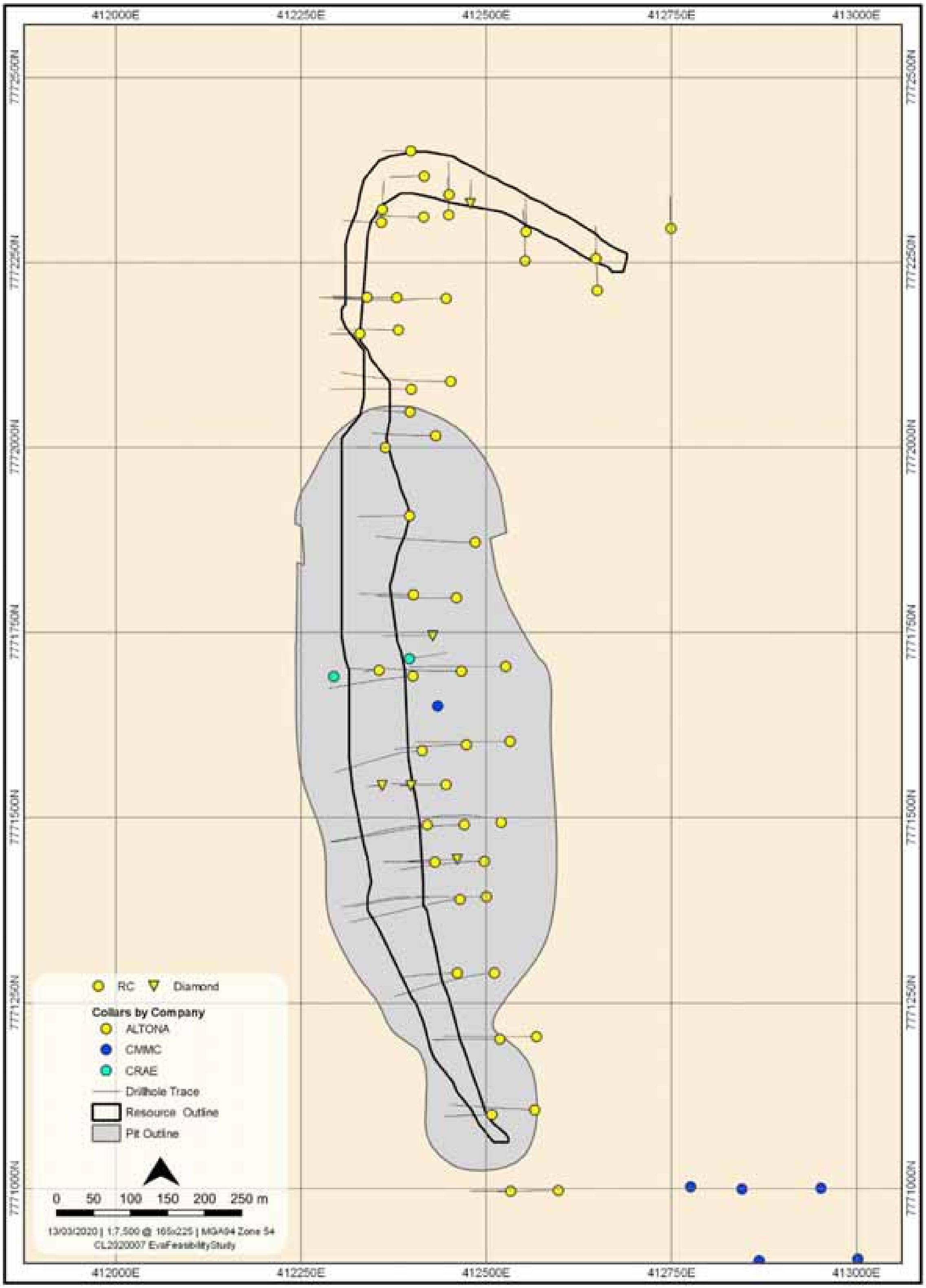


Figure 10-2: Turkey Creek Drilling Locations by Type

10.1.3 Blackard

A total of 58,388.4m of drilling in 376 holes has been completed at the Blackard deposit. Components of the drilling include 291 RC, 79 diamond, and 6 percussion drill holes completed since 1991. While early RC drill holes were relatively short and vertical, follow-up drilling was angled to keep drilling approximately perpendicular to mineralization as the deposit geometry was better understood. Drilling has been carried out relatively systematically on 50 m spaced sections, with 50 m or more tightly spaced holes along the sections. Drill holes are spaced much closer on alternating section lines (100 m spaced). A number of sections contain large step-out holes that tested for down-dip extensions of the deposit. Diamond drilling was conducted for the primary purpose of metallurgical test sampling.

The drilling history is summarized in Table 10-3, and hole locations are shown in Figure 10-3.

RC drilling typically utilized face sampling hammers (5.25", 5.5", or 6"), and diamond drilling mainly used HQ3 or NQ3 core sizes. Early rotary air blast (RAB) drilling was carried out, but these holes were not used for resource estimation.

Table 10-3: Blackard Drilling Summary

Year	Company	Hole Type	Hole Count	Metres
1991	CRAE	DD	2	411.7
		RC	1	87.0
1992	CRAE	DD	5	1,361.7
		RC	4	631.0
1993	CRAE	RC	1	100.0
1994	CRAE	PERC	6	613.0
		DD	8	1,936.0
		RC	2	302.0
1995	CRAE	DD	4	1,060.2
2002	Bolnisi	DD	7	924.8
2005	Universal	RC	121	13,558.0
		DD	19	4,081.5
		RC	81	10,563.0
2006	Universal	DD	10	1,415.0
		RC	36	3,138.0
2008	Xstrata	DD	11	4,358.4
2009	Xstrata	DD	6	2,564.1
2010	Altona	DD	4	2,324.2
		RC	7	1,687.0
2011	Altona	DD	3	548.8
		RC	20	4,028.0
2019	CMMC	RC	18	2,695.0
Total			**376**	**58,388.4**

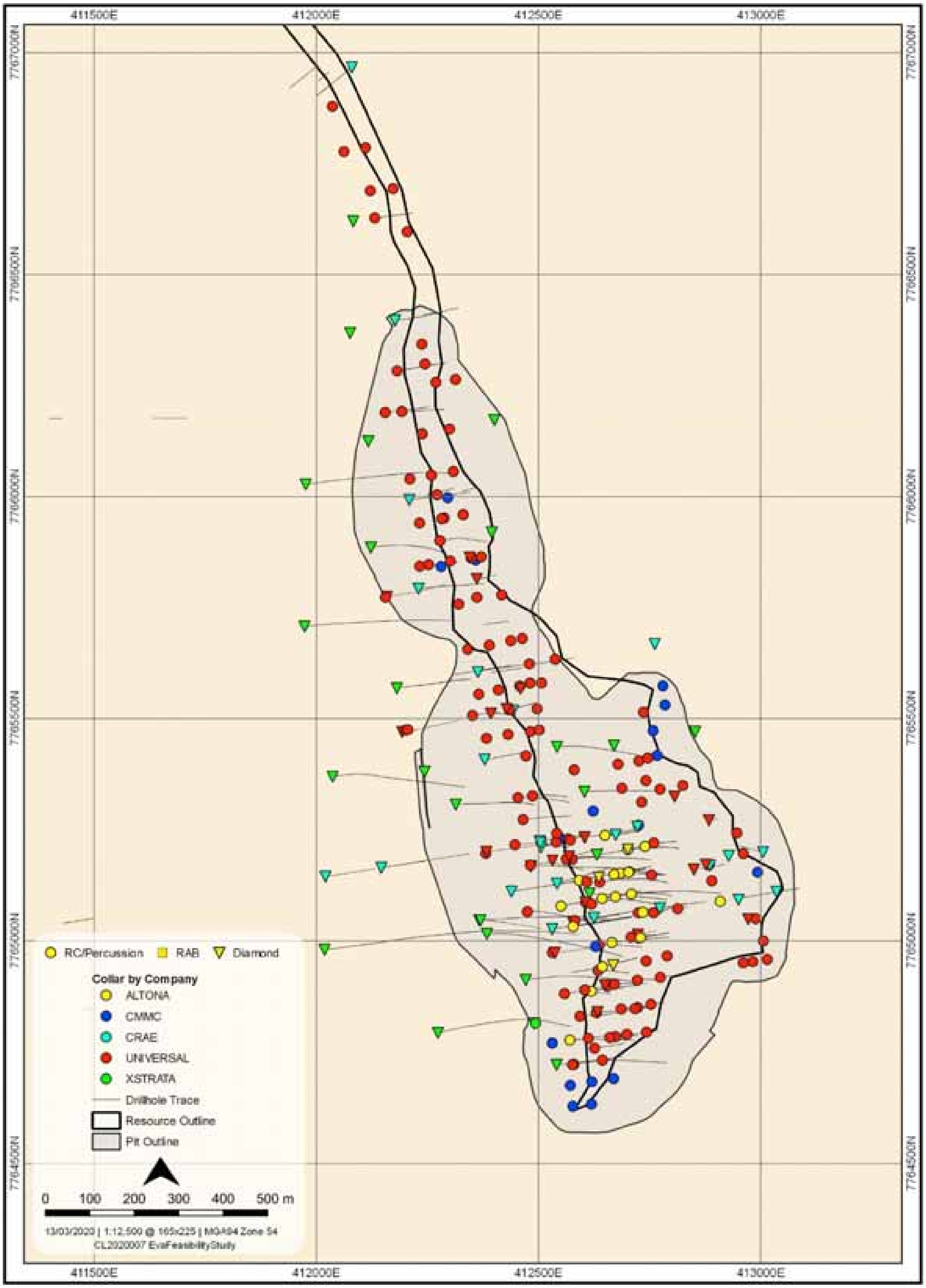


Figure 10-3: Blackard Deposit Drill Hole Locations by Type

10.1.4 Scanlan

Scanlan is a relatively near-surface deposit, and has been defined by a total of 173 drill holes for 18,979 m. Drilling is predominately RC, with only 20 of the holes being core drilling. Drill holes are either vertical or inclined, depending upon the interpreted dip of the mineralization. Drilling has been carried out on approximately 50 m spacing along 50 m spaced section lines, although alternating, or 100 m section lines, have more drill holes. In general, drill holes are more widely spaced on the northern part of the deposit, where the mineralization is narrow and vertically oriented.

The drilling history is summarized in Table 10-4, and hole locations are shown Figure 10-4. CRAE drilled 5 RC holes in 1990. Universal carried out an RAB program in 2003 as a precursor to resource-definition RC drilling from 2004 to 2009. Although the RAB holes were not used in resource estimation they did provide additional information on deposit morphology.

RC drilling typically utilized face sampling hammers (5.25", 5.5", or 6"), and diamond drilling mainly used HQ3 or NQ3 core sizes.

Table 10-4: Scanlan Drilling Summary

Year	Company	Hole Type	Hole Count	Metres
1991	CRAE	RC	24	1,086
1992	CRAE	AC	3	110
		RC	39	3,646
1993	CRAE	RC	24	1,516
1994	CRAE	RC	10	1305
		DD	5	1,403.7
1995	CRAE	DD	1	232.2
2002	Bolnisi	RC	2	397
2005	Universal	DD	9	1,594.3
		RC	45	5358
2006	Universal	DD	2	208.9
2007	Xstrata	DD	1	447
2008	Xstrata	DD	1	351.2
2010	Universal	RC	7	1324
Total			173	**18,979**

Figure 10-4: Scanlan Deposit Drill Hole Locations by Type

10.1.5 Bedford

A total of 12,240 m of drilling in 149 holes was completed at Bedford. Of these, some 68% are RC (102 holes), 30% are RAB (47 holes), and 3% are core (4 holes). RAB holes are vertical. RC and core holes were generally inclined at around -60°, drilled on 25 m spacing along 25 m spaced section lines. Section line spacing increases to 50 m and then to 100 m outside the main mineralized zones. Diamond drilling was primarily conducted for metallurgical sampling.

The drilling history is summarized in Table 10-5, and hole locations are shown in Table 10-5 and Figure 10-6. CRAE drilled 5 RC holes in 1990. Universal carried out an RAB program in 2003 as a precursor to resource definition RC drilling from 2004 to 2009.

RC drilling typically utilized face sampling hammers (5.25", 5.5", or 6"), and diamond drilling mainly used HQ3 or NQ3 core sizes. RAB drilling accounts for some 13% of drilled metres, but was not used for resource estimation.

Table 10-5: Bedford Drilling Summary

Year	Company	Hole Type	Hole Count	Metres Drilled
1990	CRAE	RC	5	420
2003	Universal	RAB	43	1,680
2004	Universal	RC	18	1,918
2005	Universal	DD	1	160
2005	Universal	RC	11	1,280
2006	Universal	DD	2	182
2006	Universal	RC	60	5,836
2009	Universal	RC	8	728
2015	Altona	DD	1	36
Total			**149**	**12,240**

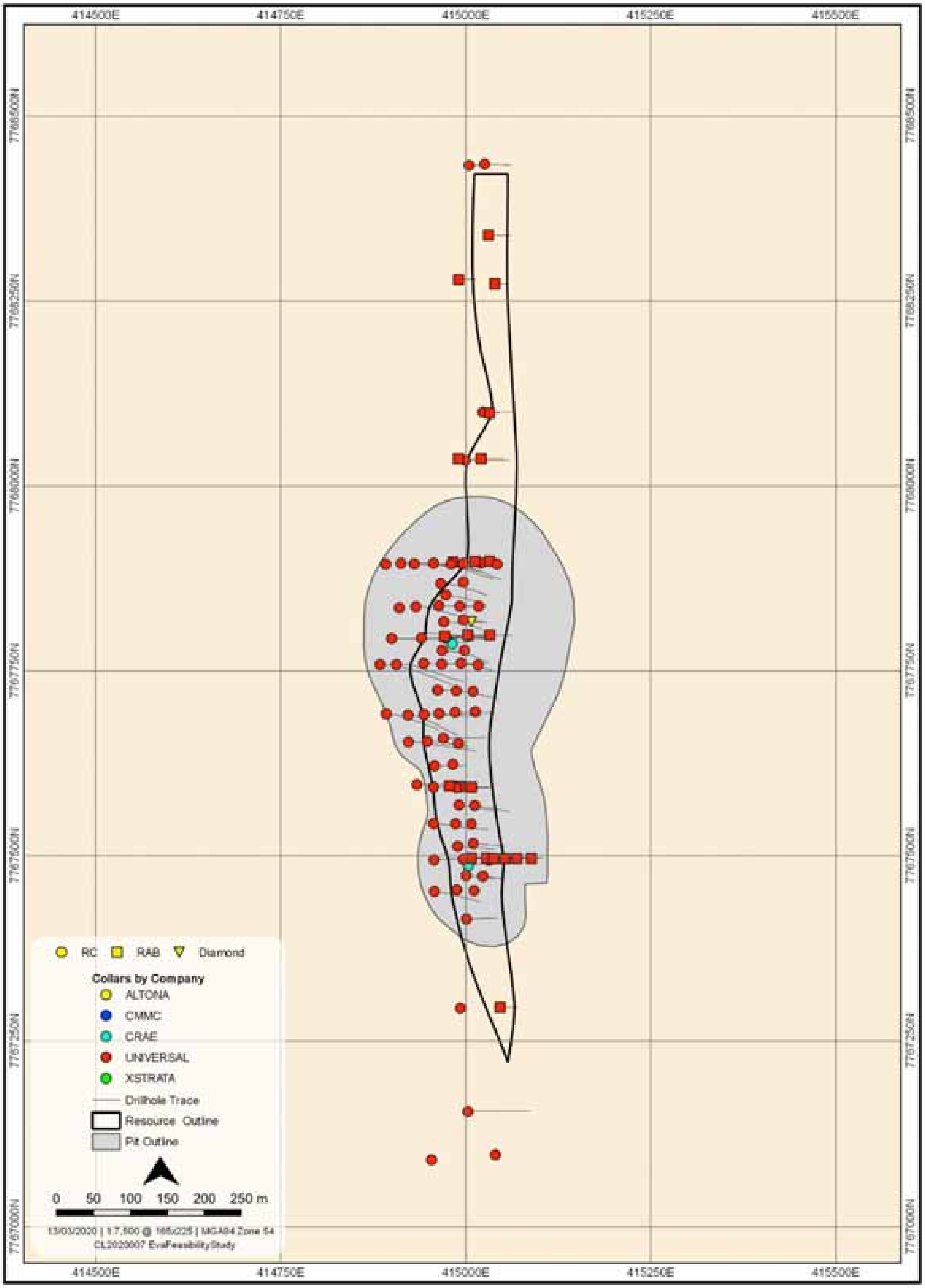


Figure 10-5: Bedford North Drill Hole Plan

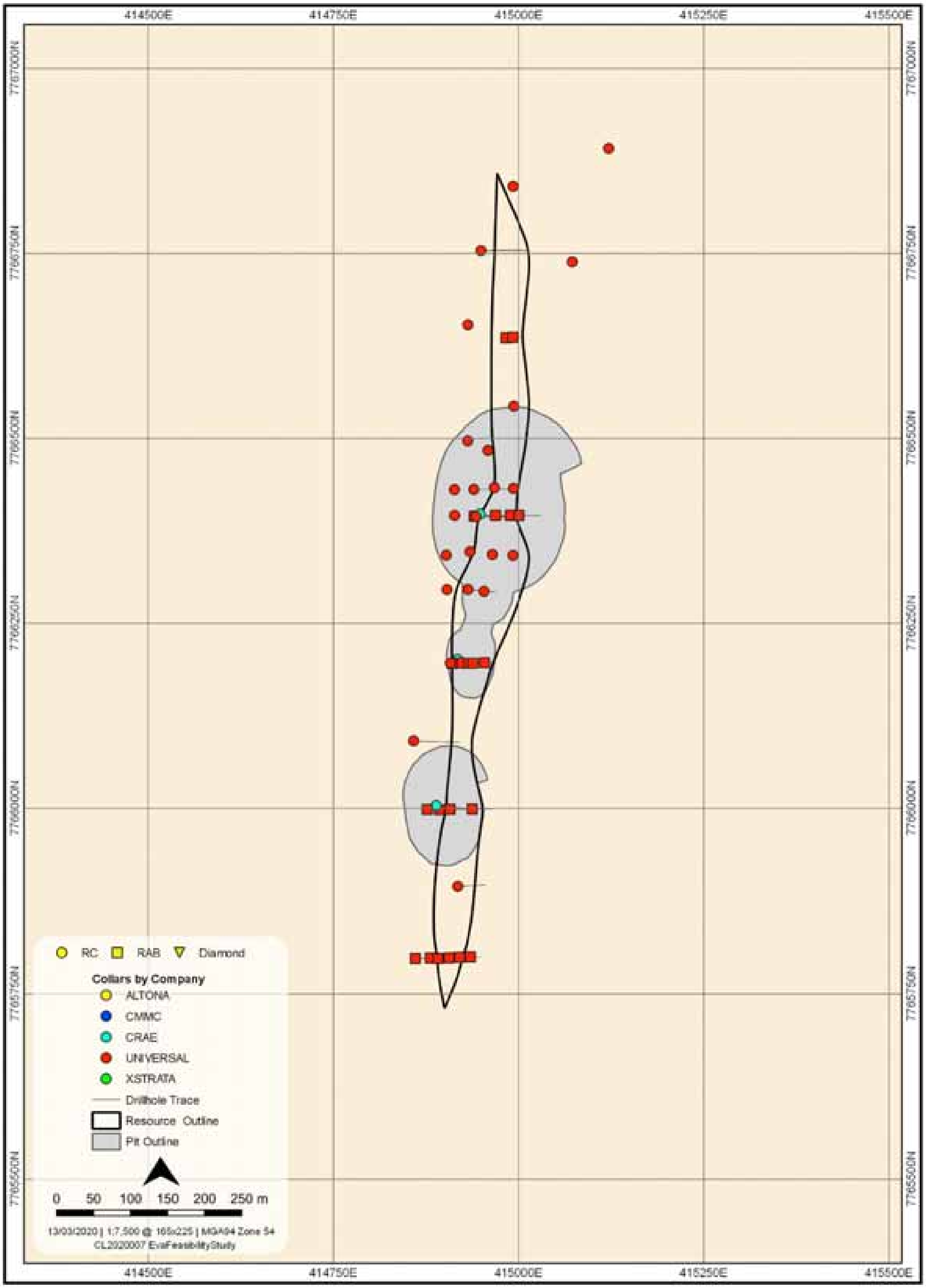


Figure 10-6: Bedford South Drill Hole Plan

10.1.6 Ivy Ann

A total of 15,145 m of drilling in 153 drill holes was completed at Ivy Ann. Of these, some 53% are RC (81 holes), 46% are percussion (PERC) (70 holes) and 1% are diamond (2 holes). Holes were generally inclined -50° to -60°, generally drilled on 50 m spaced section lines, and 20 m to 50 m along line spacing. Section line spacing increases to 100 m in Ivy Ann North.

The drilling history is summarized in Table 10-6, and hole locations are shown in Figure 10-7. Exploration on the Ivy Ann prospect began in 1992. Note that Bruce Resources became PanAust in 1995.

Table 10-6: Ivy Ann Drilling Summary

Year	Company	Hole Type	Hole Count	Metres Drilled
1992	Dominion	PERC	26	863
1992	Dominion	RC	13	1,309
1993	Dominion	RC	2	282
1995	Bruce Resources	RC	18	1,902
1996	PanAust	PERC	44	1,972
1996	PanAust	RC	3	450
1997	PanAust	DD	2	714
2003	Universal	RC	5	515
2005	Universal	RC	4	462
2006	Universal	RC	4	412
2009	Universal	RC	5	816
2011	Altona	RC	15	2,850
2012	Altona	RC	12	2,598
Total			**153**	**15,145**

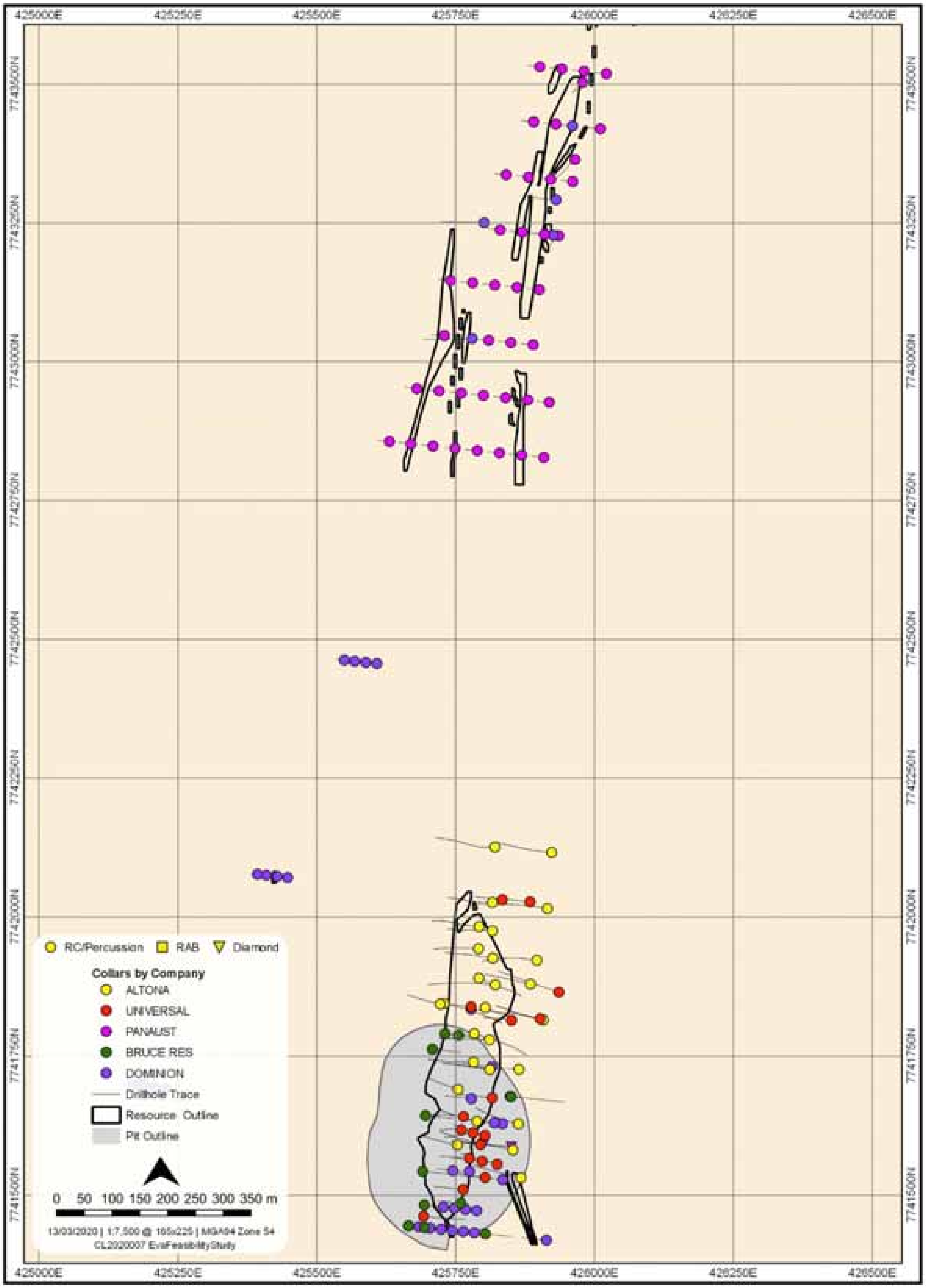


Figure 10-7: Ivy Ann Drill Collar Plan

10.1.7 Lady Clayre

A total of 25,092 m of drilling in 145 holes was completed at Lady Clayre. Of these, some 79% are RC (114 holes), 20% are diamond (29 holes) and 1% are PERC (2 holes). Holes were generally inclined -50° to -60°, generally drilled on 50 m spaced section lines, and 20 m to 50 m along line spacing.

The drilling history is summarized in Table 10-7, and hole locations are shown in Figure 10-8. Exploration on the Lady Clayre prospect began in 1978 with a single diamond hole drilled by CRAE.

RC drilling typically used 5.25", 5.5", or 6" hammers, and DDHs provided either HQ or NQ core samples.

Table 10-7: Lady Clayre Drilling Summary

Year	Company	Hole Type	Hole Count	Metres Drilled
1978	CRAE	DD	1	134
1992	CRAE	PERC	2	192
1992	CRAE	RC	11	1,188
1993	CRAE	DD	1	294
1993	CRAE	RC	9	1,250
1994	CRAE	DD	3	1,163
1994	CRAE	RC	1	102
1995	CRAE	DD	19	5,369
1995	CRAE	RC	5	464
1996	CRAE	DD	2	503
1996	CRAE	RC	10	1,484
1998	Pasminco	DD	1	180
1998	Pasminco	RC	11	1,092
2002	Universal	RC	5	1,368
2003	Universal	RC	10	1,651
2005	Universal	RC	11	1,503
2006	Universal	DD	2	154
2006	Universal	RC	11	1,353
2009	Universal	RC	3	460
2011	Altona	RC	10	1,266
2012	Altona	RC	17	3,922
Total			**145**	**25,092**

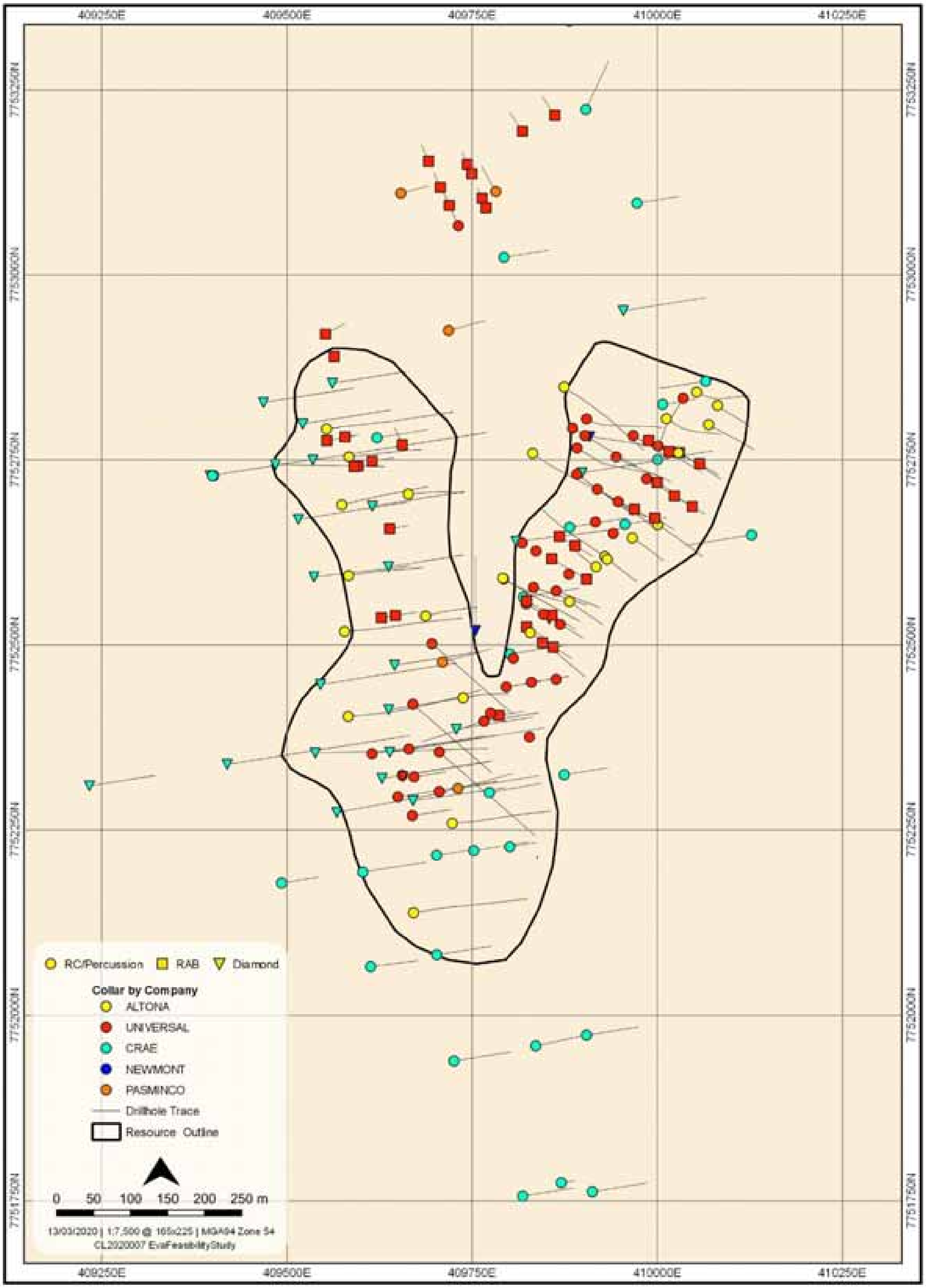


Figure 10-8: Lady Clayre Drill Collar Plan

10.2 Drill Hole Collar Survey Control

10.2.1 Little Eva

Collar coordinates for drill holes completed at Little Eva prior to 2002 were determined with reference to an informal local grid established by CRAE. In 2002, Universal resurveyed old hole collar positions at Little Eva using Differential Global Positioning System (DGPS) techniques; the work was completed by a survey contractor. In a few cases, the original collar could not be located, and earlier survey determinations by the CRAE surveyor in 1994 have been retained.

From mid-2003 through 2011, all survey work was undertaken by licensed surveyors using Trimble DGPS equipment with a minimum accuracy of ±0.05 m. All data was collected in AGD84 coordinates. From late 2011, Altona completed DGPS surveys in house using a Hemisphere R320 OmniSTAR HP GPS receiver. The system allows for real time horizontal accuracies of 10 cm to 15 cm.

Of the 523 holes drilled at Little Eva, six holes have no DGPS survey available, and the original, local grid-based, or GPS coordinate, was converted to a GPS coordinate. Geographic transformations have been used to convert original grid coordinates to GDA94 / MGA zone 54 coordinates.

10.2.2 Turkey Creek

All holes drilled by Altona, comprising the vast majority of the drilling used in defining the Mineral Resource, were surveyed with high resolution (±0.5 m) DGPS equipment. The two CRAE holes have low accuracy (±10 m).

10.2.3 Blackard

Of the 376 drill holes used in the resource estimate, 319 (85%) were surveyed by DGPS (or traditional theodolite surveys for 2 holes) with better than 0.1 m confidence. The other 57 holes were located by field GPS with an accuracy of between 3 and 10 m.

10.2.4 Scanlan

All but six of the drill holes used for resource estimation were surveyed by DGPS with better than 0.1 m accuracy. Two holes were surveyed by field GPS with accuracy of between 3 and 10 m, and another two holes have undetermined survey methods.

10.2.5 Bedford

Apart from one hole, all RC holes drilled by Universal have been located by DGPS by licensed surveyors using Trimble DGPS equipment with a minimum accuracy of ±0.02 m. All data was collected in AGD84 coordinates. The early CRAE holes (five) were initially located on local grids. Pasminco relocated the holes and recorded GPS locations for them, with a lower accuracy (±10 m). All holes drilled by Altona were surveyed with high resolution (±0.1 m) DGPS equipment.

Geographic transformations have been used to convert original grid coordinates to GDA94 / MGA zone 54 coordinates.

10.2.6 Ivy Ann

Dominion established a local grid on the prospect: all drilling carried out by Dominion and PanAust is referenced to this local grid. Universal calculated a coordinate conversion based on the locations of two early drill hole collars, and used this to transform the original local coordinates for holes drilled between 1992 and 1997 into GDA94 / MGA zone 54 coordinates.

Except for four holes drilled by Universal in 2006, all drilling completed from 2003 to 2009 has been surveyed by DGPS using Trimble DGPS equipment with a minimum accuracy of ±0.05 m. From 2011, Altona drill collars were surveyed using a Hemisphere R320 OmniSTAR HP DGPS system with horizontal accuracy of ±0.015 m.

Geographic transformations have been used to convert original grid coordinates to GDA94 / MGA zone 54 coordinates.

10.2.7 Lady Clayre

All holes drilled by CRAE from 1992 to 1994 were relocated and surveyed using DGPS by a registered surveyor in 1994. Holes drilled by CRAE, Pasminco, and Universal from 1995 to 2002 were relocated where possible and surveyed with DGPS by Universal. Survey control protocols for Universal and Altona holes are as for Little Eva.

Geographic transformations have been used to convert original grid coordinates to GDA94 / MGA zone 54 coordinates.

10.3 Downhole Surveys

10.3.1 Little Eva

All drill holes have a collar inclination and azimuth measurement in the database. The levels of hole deviation shown in Figure 10-1 are within expected ranges.

Downhole surveying of CRAE DDHs LE006 and LE076 was carried out using Eastman single shot downhole survey cameras. Survey shots were taken at approximately 40 m intervals. RC holes drilled by CRAE only have collar orientations.

Much of the RC and diamond drilling completed by Universal and Altona from 2002 to 2011 was surveyed with a variety of instruments, including those manufactured by Eastman, Camteq, Ranger, and Reflex. Survey measurements were typically taken at 40 m intervals where possible.

To overcome potential issues with the older, magnetic-based survey techniques caused by variable, and sometimes considerable, concentrations of magnetite in the rocks, Universal resurveyed all available open holes in 2005 and 2006, including those previously drilled by CRAE. A combination of a multi-shot downhole camera and a downhole gyro instrument (for magnetically quiet and active areas, respectively) was used. Multi-shot camera survey measurements are generally at 10 m intervals, and the gyro instrument surveys give semi-continuous measurements at intervals of 1 m or less. Where a hole was not open to depth, the attitude of the hole at 0 m was determined.

At the end of Altona’s 2011 program, selected holes were resurveyed using a FlexIT GyroSmart tool with readings at 5 m intervals.

From 2012 on, all Altona holes were monitored during drilling using a single or multi-shot camera, typically with completion surveys using a GyroMax isGyro.

10.3.2 Turkey Creek

All Altona holes drilled in the Turkey Creek deposit were monitored during drilling using a REFLEX EZ-TRAC camera. On completion of drilling, downhole surveys were conducted using a GyroMax isGyro, overcoming any magnetic influences inherent in the EZ-TRAC survey.

10.3.3 Blackard, Scanlan, and Bedford

The majority of the RC and diamond drilling completed by Universal and Altona from 2002 to 2018 was surveyed with downhole cameras (~69%) or gyro systems (25%), and 6% have collar orientations only. For Universal and Altona holes drilled between 2002 and present, the azimuth and inclination of the hole at the collar was measured using a compass clinometer. For the earlier holes it is unclear whether these measurements were made by survey instrument or by clinometer at the collar.

10.3.4 Ivy Ann

RC and PERC drilling completed by Dominion and PanAust only have collar orientations, and all but two of these holes are aligned along local grid directions (270° and 90°). The two DDHs completed by PanAust have downhole dip measurements, and the surface azimuths have been extrapolated down the hole. There is no record of how these dip determinations were made.

Universal and Altona used a variety of downhole camera systems to survey their drilling from 2003 to 2011. Measurements were taken at approximately 50 m intervals. Two holes only have a collar survey. In addition, Universal resurveyed selected holes in 2009 and 2011 with detailed gyro surveys. In 2009, measurements were taken at 20 m intervals, and this was reduced to 5 m in 2011. Gyro surveys were completed in 2012 on Ivy Ann drill holes completed by Altona.

10.3.5 Lady Clayre

CRAE holes drilled before 1996 only have a collar orientation. Starting in 1996, CRAE drill holes were surveyed with a downhole instrument (Eastman camera), and have at least one such measurement.

All Pasminco holes were surveyed with a downhole instrument with readings at approximately 30 m intervals, with at least one survey close to the surface, and one at the end of the hole.

Universal used a variety of downhole cameras to survey their drilling from 2002 to 2011. Measurements were taken at approximately 50 m intervals. Several holes only have a collar survey. In addition, in 2005 and 2009, Universal resurveyed selected holes with detailed multi-shot camera and gyro surveys.

All but one Altona drill hole was surveyed using a FlexIT GyroSmart tool, with readings at 5 m intervals.

10.4 Drill Hole Logging

10.4.1 Little Eva

Original hard copy drill logs or typed drilling summaries prepared by CRAE geological staff for all CRAE drill holes at Little Eva are retained in the Altona library. These are descriptive logs that were coded into the Altona system and stored in the Altona drilling database.

CRAE logged its diamond holes on variable intervals determined by lithological changes in the core. RC holes were logged on regular 1 m intervals. The early descriptive logging yielded up to two lithologies per interval, together with grain size, texture, and colour, on recoding into the digital system. Alteration and ore mineralogy were recorded as mineral species and abundance. Veining, mainly observed in core, is also logged as mineral composition and abundance. Structural and geotechnical logging of diamond holes has been done routinely from 1995 on, with orientations of veins and structures provided as dip and strike angles. The core orientation method used by CRAE is not recorded.

Universal prepared similar descriptive logs for its drilling between 2002 and 2005 that are also retained in the Altona library. RC logging was done primarily on 1 m intervals, and data was captured from these logs into the digital system in the same way as the CRAE data was captured, to provide lithology, alteration, mineralization, and veining logs. The DDH logs produced in this period were logged on intervals based on lithological changes, and included detailed structural and geotechnical logs.

Universal introduced a digital logging system based on the Surpac Logmate in 2005, and from that time on all logging has been captured digitally in coded form in the field. The templates and libraries used by this system preserved the style of logging used by both CRAE and Universal. The original digital logs produced by this system were loaded into the Altona drilling database and stored in the Altona library.

The Logmate system was replaced by Field Marshal software in 2011, and this system was used throughout the 2011 season by Altona, but the same logging procedures were followed as in prior campaigns.

In 2014, Altona completed a comprehensive lithology relogging program of available historical RC chips and diamond core. This program has provided a consistent dataset of lithology across the deposit used for resource domaining.

10.4.2 Turkey Creek, Blackard, Scanlan, and Bedford

Logging protocols, data collection and storage are all as described above for Little Eva.

10.4.3 Ivy Ann

There are no original drill logs available for the drilling completed by Dominion in 1992 and 1993, and no logging is recorded in the Altona database. RC drilling completed by PanAust in 1995 was logged on 1 m intervals, and these logs are largely descriptive. In 1996, magnetic susceptibility measurements were included. Detailed logs for the 1997 RC and diamond program contain quantitative estimates of mineralization, veining, and alteration, as well as lithological descriptions. RC holes were logged on 1 m intervals, while diamond holes were logged over intervals determined by lithology. Drill hole logging protocols for Universal and Altona are as for Little Eva.

10.4.4 Lady Clayre

Original hard copy drill logs and/or summary logs prepared by CRAE geological staff are held in the Altona library. The logs contain a description of the lithology, visual estimates of economic mineralization, and alteration. Overburden sections of RC holes and RC pre-collars on diamond holes were logged in 3 m intervals. In mineralized sections, the logging and sampling interval reduces to 1 m. Diamond holes were logged over intervals determined by the lithology, and include a graphic log of the cored sections together with structural information in the interval description. CRAE began using a lithology code during this period, which has been recoded into the Altona digital system.

Pasminco prepared logs very similar in style to CRAE for the single diamond hole and the 11 RC holes it drilled in 1998. RC holes and pre-collars were logged on 2 m intervals, while the diamond hole was logged on intervals determined by lithological changes. Logs included an uncoded lithological description, as well as visual estimates of mineralization and alteration.

Drill hole logging protocols for Universal and Altona are as for Little Eva.

10.5 Core and RC Sampling Methods

In general, sampling methodology was consistent among all deposits, but there were minor variations between the different companies and years of the program. More detailed descriptions by deposit are provided in Section 11.

Early RC sampling by CRAE used a rotary splitter mounted on the drill rig to produce 3 kg to 4 kg subsamples, which were collected in calico bags and dried on site, then sealed in polyethylene bags for shipment to the laboratory. However, in the 1994 RC sampling, CRAE used a spear to collect an approximate 3 kg sample from the cuttings. Similarly, during the 2002–2003 programs, Bolnisi employed a rig-mounted cyclone and splitter to collect 12.5% of the cuttings for dry samples, but used the spear to collect the subsample from wet cuttings. The same sampling methods were also used by Universal, Altona, and Copper Mountain Mining Corp. (CMMC) for their RC programs.

During the early programs (1991 or earlier), drill samples were collected as 3 m samples, but from 1992 onwards almost all sampling was in 2 m, or even 1 m increments.

During the later programs, beginning with Universal and then Altona, where there is better documentation, samples from RC and diamond drilling were collected and bagged in pre-numbered calico bags at the drill site during the drilling operation. Unique sample numbers were retained during the whole process. Diamond core was sawn with a diamond saw after logging, and the half core was collected as 1 m or 2 m samples and placed in the bags. RC samples were taken via a cyclone and rotary splitter mounted on the drill, producing 3 kg to 4 kg of material that was air-dried in the field. The remainder of the cuttings were bagged and laid out alongside the drill. All samples were catalogued and sealed prior to dispatch to laboratory by Altona staff. Samples were either delivered to SGS Analabs as they were collected, or stored in Altona facilities in Cloncurry prior to transport to Townsville. An extensive catalogued library of core, assay sample pulps, and RC chips are retained in the Company's Cloncurry exploration office for inspection.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

The operations and information in this section were compiled prior to Copper Mountain Mining Corp. (CMMC)'s acquisition of the Project, but have been reviewed for reasonableness and accuracy, and updated where appropriate. The sampling procedures, analytical quality, and integrity of data meet and/or exceed standards required for Mineral Reserve estimation.

11.1 Little Eva

There is very little documentary information available about sample collection and preparation for the CRA Exploration Pty. Ltd. (CRAE) drilling campaigns. The available documents covering exploration during this era lack descriptive detail when describing the mechanics of drilling and sampling procedures. The documents on the exploration work tend to assume that sampling was carried out in line with CRAE standard procedures, but these procedures are not recorded.

CRAE diamond drill holes (DDH) were sampled on approximately 2 m intervals. It is implied that the core was split or sawn and half the core retained, since the holes were later relogged by CRAE. Reverse circulation (RC) holes from LE009 to LE033 were sampled in 1 m intervals, but from LE034 to LE075 the sample interval was expanded to 2 m.

11.1.1 Universal 2002 Program

Two metre composite samples of about 2.5 kg were collected from RC chips using a modified trailer-mounted splitter. Intervals of interest were identified after the first-pass composite assays were received, and the original 1 m samples were submitted for analysis. The samples were submitted to Australian Laboratory Services (ALS) in Townsville. Sample preparation involved drying, crushing, and pulverizing the entire sample to a nominal 85% passing 75 µm. The primary analysis was by three-acid digestion followed by Atomic Adsorption Spectroscopy (AAS) for copper, and fire assay on a 30 g subsample for gold.

11.1.2 Universal 2003–2006 Program

From 2003 to 2006, Universal followed a similar procedure, except that all samples were collected on 1 m intervals using a trailer-mounted cyclone and triple-deck splitter, or similar arrangement. The major differences over the years were an increasing refinement of the QC program and a change from ALS to Analabs/SGS as the laboratory selected to do the primary analysis in 2003.

Analabs/SGS used methods that included an aqua regia digestion followed by AAS for gold, and three-acid digestion followed by AAS for copper.

The DDHs completed during this period were drilled for geotechnical and metallurgical purposes, and only the upper parts drilled with RC methods were sampled, except for two diamond holes drilled to extend RC holes that had failed to reach target depths because of poor ground conditions. Core from the extended holes was half-sawn, and samples collected in 1 m intervals for submission to Analabs.

11.1.3 Universal 2007 Program

In 2007, Universal conducted a metallurgical drill program of 10 diamond holes drilled with 1 m samples assayed at Ultratrace Laboratories, using a four-acid digestion, and analyzed by inductively coupled plasma-optical emission spectrometry (ICP-OES) for copper, and fire assay with an ICP-OES finish for gold.

11.1.4 Altona 2011 Program

For the 2011 drilling program, Altona continued with the procedures for RC sampling established by Universal, but returned to using ALS in Townsville for the primary analysis. The methods requested were ME-ICP41 (aqua regia with inductively coupled plasma-atomic emission spectroscopy (ICP-AES)) for copper, and Au-AA25 (fire assay with AAS) for gold. Copper analyses over 1% were reanalyzed with an ore grade ICP-AES method (Cu-OG46).

Two diamond holes, drilled primarily for metallurgical testing, were quarter-sawn, sampled in 1 m intervals, and submitted to ALS for analysis along with the RC samples. In addition, the core from four geotechnical holes drilled in 2005 and 2006 was recovered from the core storage, half-sawn, and submitted to ALS for analysis.

The methods requested were ME-ICP61 (four-acid digestion with ICP-AES finish), and later ME-ICP41 (aqua regia with ICP-AES) for copper, and Au-AA25 (fire assay with AAS) for gold. Copper analyses over 1% were reanalyzed with an ore grade ICP-AES method (Cu-OG46).

11.1.5 Altona-Sichuan Railway Investment Group 2015 Program

As part of a due diligence of the Project assets, Sichuan Railway Investment Group (SRIG) drilled two confirmatory triple-tube diamond holes at Little Eva using HQ core. An independent consultant for SRIG managed the program. The holes were submitted to ALS for cutting (half core) and analysis.

Altona drilled two diamond holes for metallurgical testwork. These were quarter-sawn and sent to ALS Perth to be assayed using ME-MS41 (aqua regia with ICP-MS) for copper and Au-AA25 (fire assay with AAS) for gold. Copper analyses over 1% were reanalyzed with an ore grade ICP-AES method (Cu-OG46).

Four DDHs, drilled for geotechnical purposes in 2005 and 2011, were half-sawn and submitted to ALS for analysis. The methods requested were ME-ICP41 (aqua regia with ICP-AES) for copper, and Au-AA25 (fire assay with AAS) for gold. Copper analyses over 1% were reanalyzed with an ore grade ICP-AES method (Cu-OG46).

11.1.6 Work Subsequent to Resource Calculation

Since completion of the Little Eva Resource, Copper Mountain completed eight additional HQ-sized diamond core holes within the prospect area for geotechnical and metallurgical studies.

11.1.7 Quality Control Procedures

The QC procedures employed by CRAE are poorly recorded, and appear to have been at a low level by modern standards. For the programs from 2002 onwards, Universal implemented quality control programs which meet with currently accepted practices, and included field duplicates, triplicates, reference standards, and blanks. No problems within the resource data were revealed by the quality assurance and quality control (QA/QC) program.

The data quality and QC procedures were reviewed in December 2009 in the Independent Mineral Specialist Report prepared by Optiro (Glacken, 2009). Optiro noted that good industry QA/QC practices were applied, with reasonable rates of inserted standards, repeats, and blanks.

For the 2011 drilling program, Altona continued with the QC procedures established by Universal in 2006, which included:

- Regular duplicate sampling of RC cuttings at a rate of 1 in 20 primary samples.
- Triplicate samples collected at the time of drilling at a rate of 1 in 40 primary samples, submitted to an umpire laboratory.
- Submission of Certified Reference Materials (CRMs) or standard samples at an overall rate of 1 in 20.
- Submission of blank samples at an overall rate of 1 in 45 primary samples.

11.2 Turkey Creek

All RC drilling between 2012 and 2014 was completed using either a 140 mm or 5.5" hammer drill. RC chips were collected at 1 m intervals, as per Altona Mining's standard procedures.

QA/QC protocols for the 2012 and 2014 drilling programs at Turkey Creek included the insertion of CRMs at a ratio of 1 in 20. Field duplicates were taken from the RC drilling using a riffle splitter on site, also at 1 in 20 rates. All samples were sent to ALS Townsville, and a standard sample protocol of drying, crushing, splitting, and pulverizing was followed, resulting in 250 g pulp samples. These were submitted for ME-MS41 (aqua regia digestion with ICP-MS finish). The aqua regia digestion dissolves sulphide and oxide minerals, but does not dissolve silicates, so the copper contained in the hydrobiotite will not be reported. Copper analyses over 1% were reanalyzed with an ore grade ICP-AES method (Cu-OG46). Gold was determined via Au-AA25 (fire assay with AAS).

In 2015, Altona drilled five diamond holes for metallurgical samples. Core from these holes was sent to ALS Ammtec in Perth, where whole core samples were taken at 1 m intervals and assayed using ME-MS41 (aqua regia with ICP-MS) for copper, and Au-AA25 (fire assay with AAS) for gold. Copper analyses over 1% were reanalyzed with an ore grade ICP-AES method (Cu-OG46).

11.3 Blackard and Scanlan

Early campaigns of diamond drilling by Bolnisi and CRAE at Blackard and Scanlan produced core of various sizes, including 4.5", 5.375", NQ, NQ2, HQ, and HQ3. Half-core or quarter-core samples were routinely cut at intervals of either 1 m or 2 m.

RC drilling by CRAE and Bolini was predominantly drilled with a 130 mm diameter hammer drill. Percussion drilling by CRAE was completed using either a 4.5" or 5.5" hammer drill. Chip samples were collected on either 1 m, 2 m, or 3 m intervals using standard CMMPL procedures.

Samples submitted by CRAE and Bolnisi were typically assayed by Analab using either four-acid digestion (hydrofluoric, perchloric, hydrochloric, and nitric) with an AAS finish, or aqua regia digestion with an ICP-OES finish.

Diamond core drilled by Universal and Xstrata at the Blackard and Scanlan deposits were typically either of NQ or HQ3 diameter, and routinely sampled as either half or quarter core at either 1 m or 2 m intervals within mineralized domains. Material drilled in the barren hanging wall was cut as either half or quarter core at intervals of up to 6 m.

RC drilling completed by Universal and Xstrata typically utilized a 5" hammer drill, with samples collected at either 1 m or 2 m intervals, as per standard CMMPL procedures.

Samples were typically submitted by Universal and Xstrata to either SGS, Analabs, or ALS Townsville (or ALS Mount Isa) for either:

- ME-ICP41 (trace level analysis of 34 elements by aqua regia digestion with ICP-AES finish)
- MEMS-61 (ultra trace level analysis of 47 elements by four-acid "near total" digestion [HF-HNO_3-$HClO_4$ acid digestion, HCl leach] and a combination of ICP-MS and ICP-AES finishes)
- Hot aqua regia digestion, diluted HCl added to residue, with an AAS finish
- Cu-OG46 ore grade copper analysis by aqua regia digestion, with either AAS or ICP-AES finish.

Diamond core drilled by Altona Mining for metallurgical purposes was typically drilled with a HQ3 bit at 1 m intervals, and sawn to quarter core.

RC drilling by Altona Mining was completed with a 5.5" hammer drill and sampled at 1 m intervals using standard Altona procedures as outlined for Little Eva.

Samples were submitted by Altona Mining to ALS Townsville for either ME-ICP41 (trace level analysis of 34 elements by aqua regia digestion with ICP-AES finish), or Cu-OG46 (ore grade copper analysis by aqua regia digestion, with either AAS or ICP-AES finish).

CMMC completed 18 RC drill holes in 2019 at Blackard with a 5.75" hammer drill. Samples were collected using standard CMMPL procedures at intervals of 2 m.

Samples were submitted by CMMPL to ALS Townsville for either ME-ICP61 (trace level analysis of 27 elements by four-acid "near total" digestion [HF-HNO_3-$HClO_4$ acid digestion, HCl leach] and an ICP-AES finish) or Cu-OG62 (ore grade copper analysis by HF-HNO_3-$HClO_4$ digestion, HCl leach for use as over-range, with either AAS or ICP-AES finish).

Early QC procedures used by CRAE consisted of duplicate samples (1 in 15) and repeat assays (1 in 15), but insertion of blanks or standards into the sample stream was not documented. Comparison of sample and analytical duplicates raises no concerns. Bolnisi implemented a QC protocol for their drill programs by using field duplicates at the rate of 1 in 50 and inserting native copper standards at the rate of 1 in 40. It is assumed that the native copper standards were used due to potential problems during assaying, which may have included the potential for native copper to smear on grinding plates and contaminate subsequent samples, and segregation of metallic particles during processing yielding poor reproducibility. From 2005 on, Universal implemented a QC program for RC drilling that used CRMs (1 in 30), field duplicates (1 in 20), and blanks (1 in 40). Results indicated that variability of assay data in the native copper zone is significant in a modest number of the samples, and therefore use of an umpire laboratory check at the rate of 1 sample in 40 was implemented in 2004.

Universal designed sampling and specific analytical protocols for oxide or native copper, samples, and sulphide zone drill programs. These protocols have been maintained or only slightly modified since that time. The sampling for oxide and primary mineralization is the same, using a trailer-mounted cyclone and triple-deck splitter that divides the RC cuttings into 12.5% and 87.5% volume splits. The larger sample is stored on site in plastic bags. Subsamples are collected from the larger split, on every tenth sample in the native copper zone, and every 20th sample in the sulphide zone, and inserted into the sample shipment stream. Additionally, a sequence of CRMs and blanks are inserted into the sample stream at the rate of 1 in every 40 samples. Finally, a second subsample is collected from the larger split, at a frequency of 1 in 30 for the native copper zone, and 1 in 40 for the sulphide zone, and shipped to a second laboratory.

The analytical protocol for the sulphide analysis is as follows: oven-dry entire sample and pulverize to 85% passing 75 µm, then remove a 1 g subsample with a duplicate sample at the rate of 1 in 20; insert blank and reference samples into the sample stream, each at the rate of 1 in 50; use three-acid digestion, and analyze for copper by AAS.

The analytical protocol for the native copper zone samples is more involved. Samples are oven-dried and then weighed, jaw crushed to -6 mm, then ground in a disc mill (Analabs Supercrunch) to -500 µm. One in 20 samples are reweighed to check for weight loss. Riffle-split into a 1 kg subsample and residual. A duplicate sample is taken from every 20th residual sample. Subsamples are pulverized to P_{85} 75 µm in a ring mill. A 20 g split is taken, with another duplicate at 1 in 20. Blanks and reference standards are inserted at a rate of 1 in 50. Aqua regia digestion is used, and analyzed by AAS.

11.4 Bedford

Sampling and QA/QC protocols for Bedford are as for Little Eva, except during 2009; the sampling procedure employed by Universal in 2009 was essentially unchanged from their earlier work. Universal initially used a 6" hammer drill, then later a 5.375" hammer drill for RC drilling. The majority of samples were collected at 1 m intervals, with a small number of early samples collected at 2 m intervals using standard Universal procedures. Universal drilled several diamond holes with either NQ3 or HQ3 core diameter. This core was cut to half- or quarter-core subsamples for laboratory submission.

Early Universal sampling was submitted to Analabs Townsville for mixed acid, ore grade AAS analysis (old code GA145). Later sampling was submitted to SGS, with methods modified to include a multi-element ICP-OES method (ICP21R) for Ag, Al, As, Ba, Bi, Ca, Cd, Co, Cr, Cu, Fe, K, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Se, Sn, Sr, Ti, U, V, Zn, and Zr. Gold was determined by method FAA505 (50 g fire assay, followed by AAS).

In 2015, Altona drilled one diamond hole for metallurgical testwork that was quarter-cored and sent to ALS Perth to be assayed using ME-MS41 (aqua regia with ICP-MS) for copper, and Au-AA25 (fire assay with AAS) for gold. Copper analyses over 1% were reanalyzed with an ore grade ICP-AES method (Cu-OG46).

11.5 Ivy Ann

The sampling procedures used by Dominion in 1992 and 1993 are not recorded, but all RC and PERC drill holes were sampled on a uniform 2 m interval and analyzed for copper and gold. From 1995 to 1996, all RC holes were sampled on 2 m intervals and riffle-split to produce a nominal 4 kg sample for analysis. Samples were dispatched to ALS for analysis for copper and cobalt using method G001 (perchloric acid digestion followed by flame AAS), and using method PM203 for gold (fire assay with AAS). Approximately 1 in 20 samples were resampled at the drill as field duplicates, but there is no report or evidence that CRMs or blanks were used in the program. In 1997, the analytical method for base metals was changed to ICP method, and the suite was extended to include Pb, Zn, As, Ni, and Mo.

The two-DDHs completed in 1997 were sampled on 1 m intervals, and submitted to ALS for assay for the same elements as the RC drilling.

RC drilling completed by Altona in 2012 utilized a 140 mm hammer drill, with samples collected at 1 m intervals, as per standard Altona procedures.

Altona submitted samples to ALS Townsville for analysis by ME-MS41 (aqua regia with ICP-MS) for copper, and Au-AA25 (fire assay with AAS) for gold. Copper analyses over 1% were reanalyzed with an ore grade ICP-AES method (Cu-OG46).

11.6 Lady Clayre

All CRAE diamond holes from 1992 through 1996 were sampled on 1 m intervals. CRAE RC drilling and RC pre-collars on diamond holes for the 1992 campaign were routinely sampled in 3 m intervals in non-mineralized sections and pre-collars. Mineralized sections were sampled on 1 m intervals. In later years, CRAE standardized to 2 m intervals for all RC holes. Details of the laboratories and analytical procedures used are not recorded.

Pasminco drilled one RC/diamond hole and 11 RC holes into the Lady Clayre prospect in 1998. The RC sections were sampled in 2 m intervals, and the diamond sections were sampled in 1 m intervals. Samples were analyzed by Amdel Analytical laboratories (Amdel) using fire assay/AAS for gold, and mixed acid/ICP-OES for copper and base metals.

RC drilling completed by Altona in 2012 utilized a 140 mm hammer drill with samples collected at 1 m intervals, as per standard Altona procedures.

Altona submitted samples to ALS Townsville for analysis by ME-MS41 (aqua regia with ICP-MS) for copper, and Au-AA25 (fire assay with AAS) for gold. Copper analyses over 1% were reanalyzed with an ore grade ICP-AES method (Cu-OG46).

11.7 Security

Samples from RC and diamond drilling programs were collected and bagged into pre-numbered calico bags at the drill site during drilling operations. Unique sample numbers were retained during the entire project. All samples were then catalogued and sealed prior to dispatch to laboratory or secure storage facilities by Altona staff. Samples were either collected daily and delivered to Analabs/SGS, or delivered to and stored in Altona facilities in Cloncurry prior to shipment to laboratories in Townsville.

A catalogued and extensive library of core, assay sample pulps, and RC chips is retained in the Company’s Cloncurry exploration office for inspection.

12 DATA VERIFICATION

Estimation of Mineral Resources and Mineral Reserves relies on analytical data (assays) from samples collected from drill holes, and the position of those samples in 3D space. The methods and quality of the sample collection procedures and analytical data were examined previously and reported on by independent consultants. Additionally, data validation and verification has been undertaken by Copper Mountain Mining Corp. (CMMC). Physical verification of drill hole locations and additional drilling was only completed on the Little Eva, Turkey Creek, and Blackard deposits, the four largest deposits. The quality of the assay databases was investigated for all deposits but primarily focused on these four largest deposits.

Altona maintained a very extensive and high-quality database using Datashed software and has carefully preserved historical records and thoroughly documented checks and resurveys of drill hole collar locations and downhole surveys. All six drill collars checked in the field with handheld Global Positioning System (GPS) units on the Little Eva deposit were found to be correctly positioned. Review of drill holes on section did not reveal any anomalies with respect to drill hole locations or deflections. Checking the database against analytical certificates for approximately 200 samples did not reveal any discrepancies, and confirmed placement of standards and blanks into the sample stream. Visual examination and estimation of copper grades in drill core and cuttings at Altona's core storage yard was consistent with recorded analytical data. Previous checks by third-party consultants, including SRK and Optiro, reported similar satisfaction with data quality.

Statistical analysis of the Project drill data separated by company and/or year of drilling, as reported in Section 14, indicates that there is no systematic bias to the data, either by company or drill type. Assaying at the Copper Mountain Mine from two drill holes drilled in the Little Eva starter-pit area to collect material for metallurgical testing closely matches block grades within the resource block model, providing additional validation of the dataset and estimation methodology. Additional drilling carried out on the Turkey Creek and Blackard deposits by CMMC conformed to previous results.

It is concluded that the databases for all deposits of interest are suitable for use in resource estimation.

13 METALLURGICAL TESTING

13.1 Introduction

This section summarizes both historical and recent testwork associated with the various ore types on the Project property. For additional information, please reference the 2018 Feasibility Study completed by Hatch for CMMC in 2018, the GR Engineering Services (GRES) Definitive Feasibility Study (DFS) for Altona in 2014, and the GRES DFS for Universal in 2009. The previous feasibility studies discuss in detail the metallurgical performance of ores from the Little Eva pit and associated satellite pits, which contain classic, flotation-amenable copper sulphide ore types. Work completed as part of the present feasibility study expands upon the previous feasibility studies and considers the addition of other pits, including those containing native copper-bearing ore, which require more unique processing approaches, as had been the focus of the earlier 2009 DFS. This report generalizes the various ore sources into one of two classes for design purposes: sulphides, and native copper. The various ore sources were studied from the perspective of newer technologies, including high pressure grinding rolls (HPGR) for comminution, and direct flotation reactors for flotation.

The Little Eva pit is the main ore source for the Project, containing 97.7 Mt at 0.38% Cu and 0.07 g/t gold sulphide ore. This pit has been well studied, with 145 flotation tests from multiple core and RC chip sources that ranged in scope from benchtop to pilot plant. This ore consistently demonstrates high recovery performance with a high degree of liberation at relatively coarse grinds. The average ore competency lies near the 50th percentile of the JK database, with medium to hard Bond work indices. Copper is present as chalcopyrite with trace amounts of pyrite. Strong flotation kinetics result in high recoveries, concentrating to a saleable final concentrate grade following a nominal regrind with no pH modification. The gold is predominantly associated with the chalcopyrite and reports to the copper concentrate. Overall, this ore type presents low technical risk.

The sulphide satellite pits, comprising Turkey Creek, Bedford, Lady Clayre, and Ivy Ann, are smaller sources, together representing 19.4 Mt of ore. These ore types are generally similar to Little Eva from both a comminution and flotation perspective. Some differences include a stronger deportment of copper to bornite and varying grade distribution. Overall, these pits show average copper recoveries of 88% to 95%, and represent high-grade sources of high recovery material. The specific recoveries for each pit are used as inputs into the mine schedule and financial model.

The native copper-bearing pits, Blackard and Scanlan, are distinctly different from other pits in the area, containing oxide cap, native copper, sulphide transition, and sulphide zones. Combined, these pits represent 53.8 Mt of ore. The native copper zones are the largest copper-bearing zones within these pits, containing a relatively fine distribution of native copper with varying quantities of sulphides. These pits were studied by previous owners; however, several recent updates have been completed. In total, 410 flotation tests (including blended ore feed) have been completed, ranging from benchtop to pilot scale work. On a flotation basis, the native copper zones typically achieve 60% recovery, with an additional 2% to 3% achievable by gravity methods. Recovery is highly variable as deportment shifts from native copper to sulphides, requiring flexibility within the processing flowsheet between gravity and flotation operations to achieve an average of 63% overall native copper recovery. This ore is typically very soft, resulting in low comminution costs and high mill throughputs. Below the native copper-bearing zones of both Blackard and Scanlan are sulphide zones containing bornite and chalcopyrite, behaving similarly to Turkey Creek ore. The flotation response of the ore from the native copper to the sulphide transition zone increases with sulphide content, as expected.

For determining key comminution values for plant design, the 70th percentile of the dataset was used to ensure confidence in comminution equipment sizing. For this feasibility study, Ausenco's proprietary Ausgrind power-based calculation suite was used, which is mainly driven by Dr. Steve Morrell (SMC Test®) parameters and Bond work indices (Lane et al., 2013).

In total, the abovementioned work is sourced from 25 metallurgical testing campaigns completed at established metallurgical labs throughout Australia and British Columbia, Canada, from 1996 to 2019.

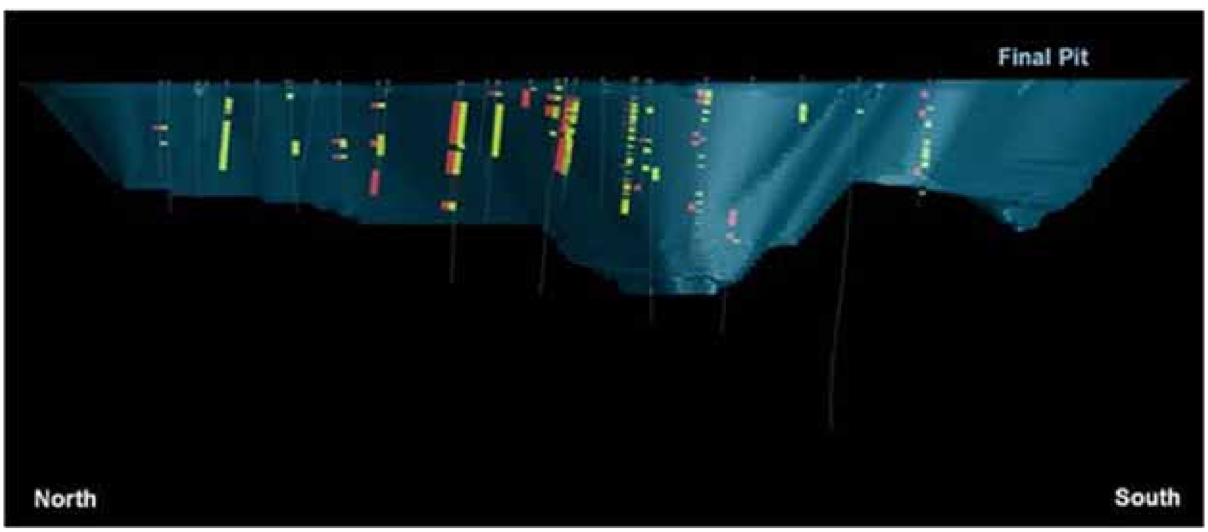
13.2 Little Eva Deposit

The Little Eva pit is classified as an iron oxide copper gold (IOCG) deposit. Copper is present as chalcopyrite, with trace amounts of bornite and chalcocite. The host rock contains high levels of iron oxides such as hematite and magnetite. Most of the deposit contains trace quantities of pyrite requiring no pH modification at the cleaner stage. Chalcopyrite is present in relatively coarse grain sizes, resulting in 95% liberation at 212 µm. Overall, this ore presents minimal challenges from a metallurgical perspective, as it has average comminution characteristics and yields high copper recovery.

Samples for both comminution and flotation testing were gathered from various locations and depths to gain a solid understanding of the LOM performance. Multiple holes were selected that were subsequently used in several metallurgical studies in the form of bulk composites, and individual continuous core lengths.

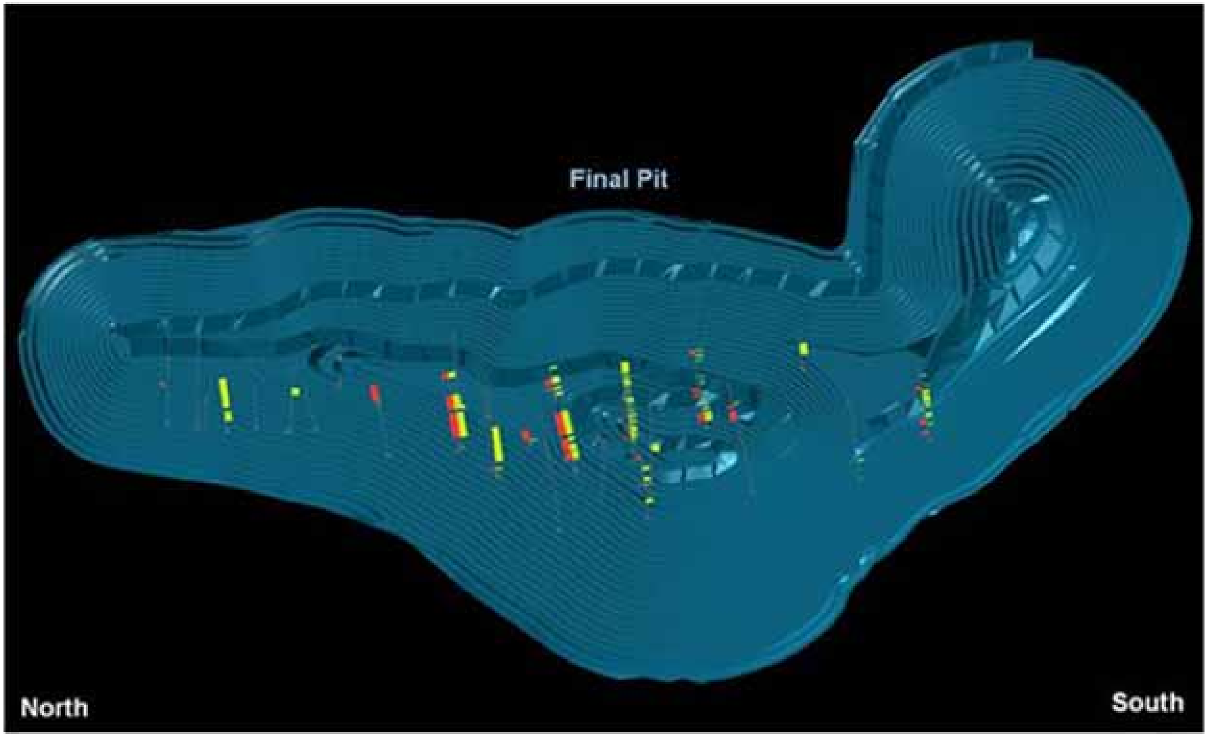
Table 13-1: Little Eva Pit Sample Summary

	Flotation	Comminution
# of Holes Sampled	24	13



Notes: Red = Comminution testwork. Yellow = Flotation testwork. Projection looking east.

Figure 13-1: Distribution of Composite Samples used for Metallurgical Testing of the Little Eva Deposit showing Final Pit



Notes: Red = Comminution testwork. Yellow = Flotation testwork.

Figure 13-2: Distribution of Composite Samples Used for Metallurgical Testing of the Little Eva Deposit – Plan View

13.2.1 Mineralogy

Previous mineralogical studies of the Little Eva pit highlighted that the ore is predominantly feldspar, quartz, carbonate, amphibole, biotite mica and iron oxide minerals, with minor to trace amounts of copper and iron sulphides. The deposit is low in overall sulphur content, with sulphur assays commonly being less than 0.8% but ranging as high as 1.6%. QEMSCAN analysis of the bulk flotation feed and the tailings composite samples has identified chalcopyrite ($CuFeS_2$) as the main copper-bearing mineral, locally ranging in abundance from 0.1% to 2%. Trace bornite (Cu_5FeS_4) usually occurs intergrown with chalcopyrite, and is less than one tenth the abundance of chalcopyrite. Pyrite (FeS_2) and chalcocite (Cu_2S) occur in ultra-trace amounts of about one hundredth the abundance of chalcopyrite. A scanning electron microscope (SEM) analysis of hand-panned flotation concentrate identified very fine particles (ranging in size from 2 µm to 9 µm) of electrum (gold ± silver) associated with pyrite and/or chalcopyrite. Figure 13-3 and Figure 13-4 show typical chalcopyrite and bornite associations with gangue minerals within the host rock.

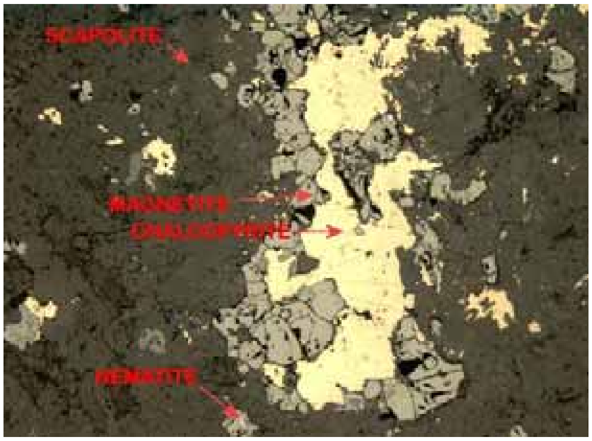


Figure 13-3: Drill Hole LED495, Specimen 94975, Scale 4.6 mm

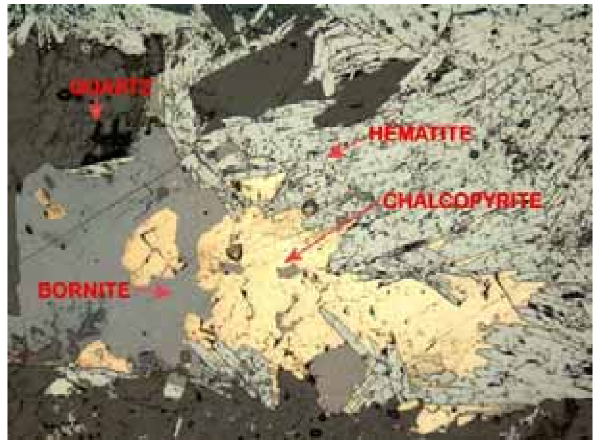


Figure 13-4: Drill Hole LED495, Specimen 94966, Scale 1.6 mm

13.2.2 Little Eva Comminution

Since 2005, 44 unique samples have been processed for comminution characterization. This includes both bulk composites and specific smaller diamond drill core intervals. Test results provided information for Bond work indices and SMC parameters that were subsequently used for mill sizing.

Most of the Little Eva dataset lies near the 50th percentile of the JK database in terms of Axb values.

	Average	70th Percentile
SG	2.81	2.87
Axb	47	39
RWi	19.6	20.6
BWi	17.0	18.7

When plotted spatially, ore hardness tends to be softest in the northern portion at the starter pit location, with Bond ball mill work indices (BWi) averaging 15.5 kWh/t. The ores then tend to be harder moving south and deeper into the pit, peaking at a BWi of 20 kWh/t. In most cases, BWi values were determined using a closing screen of 212 µm.

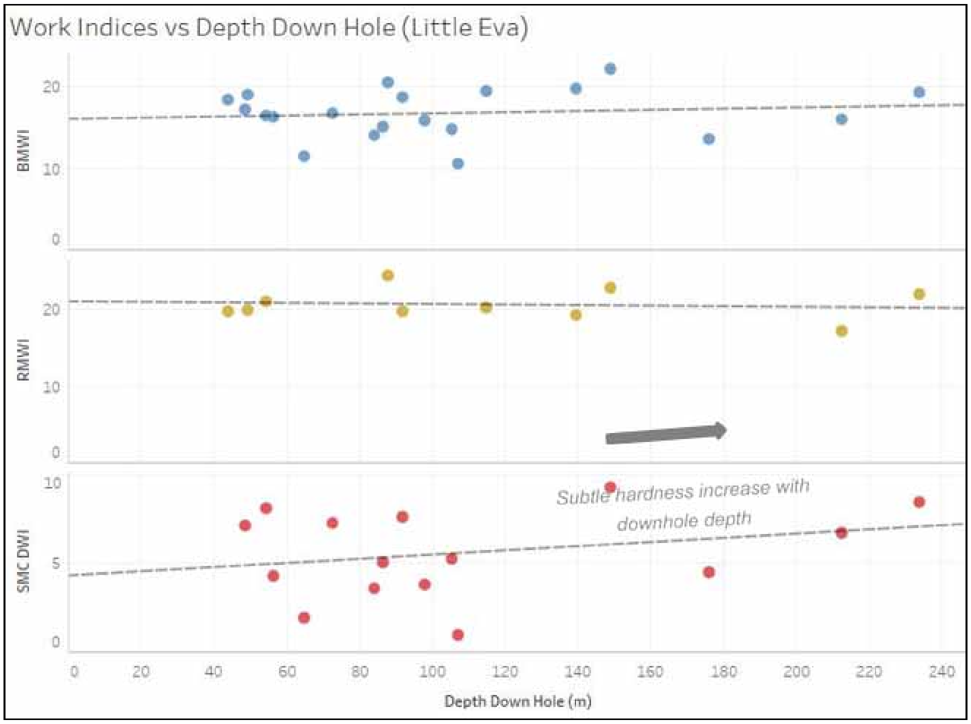
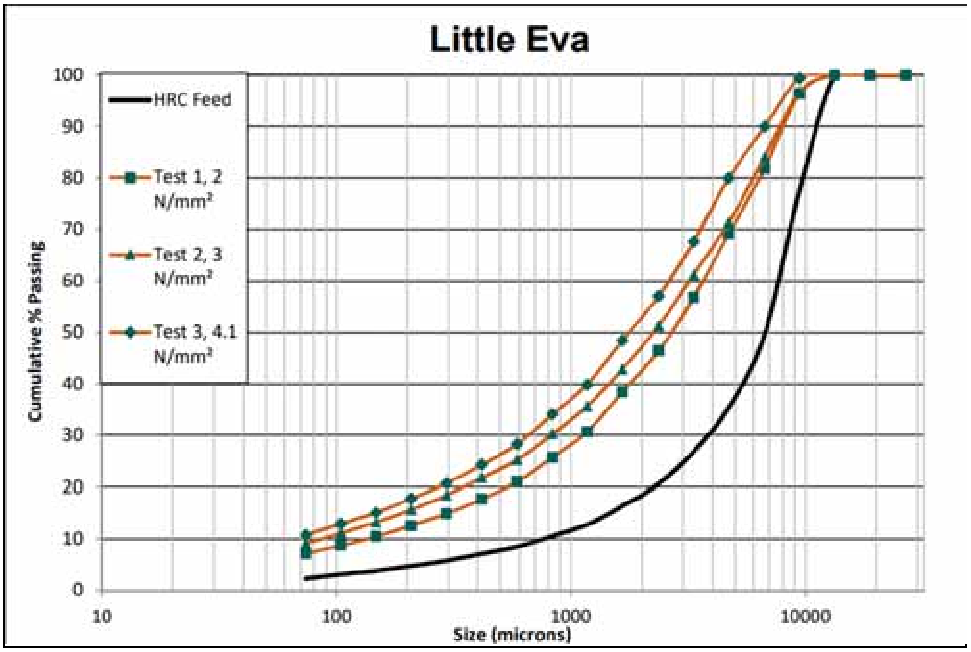


Figure 13-5: Ball Mill Work, Rod Mill Work, and SMC Drop Weight Indices vs. Little Eva Northing

This feasibility update included the change from a traditional SAG and ball milling with pebble crushing circuit (SABC) to a stage crushing circuit, featuring an HPGR in tertiary stage, followed by ball milling (2C-HPGR-B). Although HPGR formats typically favour more competent ore types, the cost structure in north Queensland supports minimizing power consumption and reducing the consumables and labour associated with SAG mill relines. Fresh samples of Little Eva core were sent to the Metso York Laboratory for packed bed and single pass HRC300 testwork.



Source: Metso, 2019

Figure 13-6: HRC300 Single Pass Test Results for Little Eva Ore

The results highlighted no concerns in terms of product sizing or specific throughput. The ore was classified as medium for abrasiveness. Metso scaled-up specific throughput for the Little Eva ore was 311 tonnes seconds per cubic metre hour (ts/m^{3}h). These results were used to estimate a design specific throughput of 291 ts/m^{3}h when blended with Blackard material (see Section 17).

13.2.3 Little Eva Flotation

Since 2005, 140 flotation tests have been performed on ore from the Little Eva pit. The scope of these tests has ranged from bench top flotation work through to full pilot plant studies. All flotation campaigns, excluding oxide ores, have yielded recovery results. Recent testing on diamond drill core samples completed by Copper Mountain Mining Corp. (CMMC) in 2019 has provided results in line with historical values, as shown in Figure 13-7.

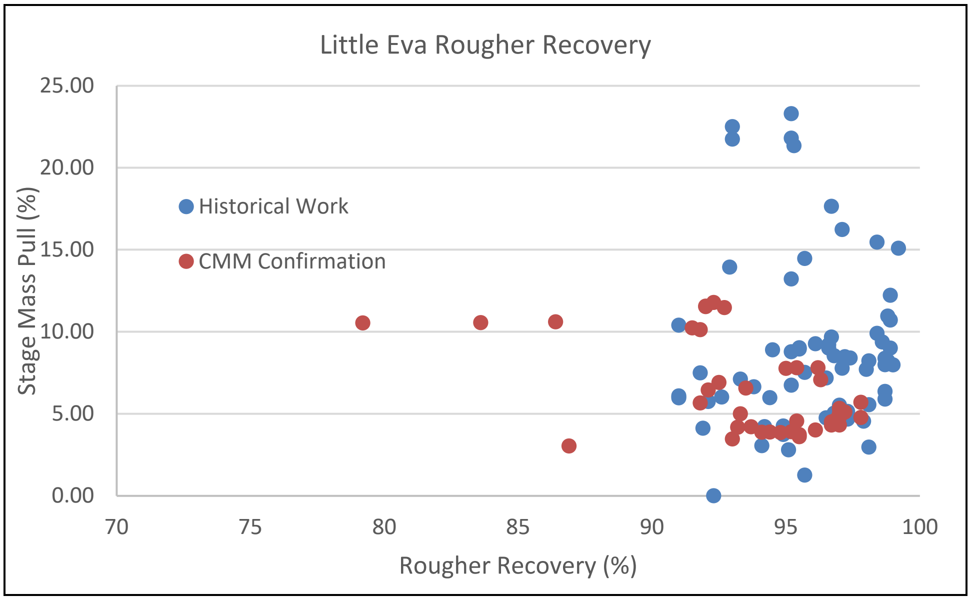


Figure 13-7: Little Eva Rougher Copper Recovery vs. Mass Pull

The target copper recovery for ores from Little Eva has varied from study to study as the milling equipment and grind conditions have changed. The two main parameters for assigning a target recovery are grind (liberation), grind size (ranging from 106 µm up to 250 µm), and grade.

The Little Eva pit mine plan contains years where copper content in mill feed averages between 0.299% Cu and 0.526% Cu. Generally, there was no significant decrease in average rougher flotation recovery as a function of grade over this range, as shown in Figure 13-8, which depicts all available rougher flotation results, though not filtered by grind size.

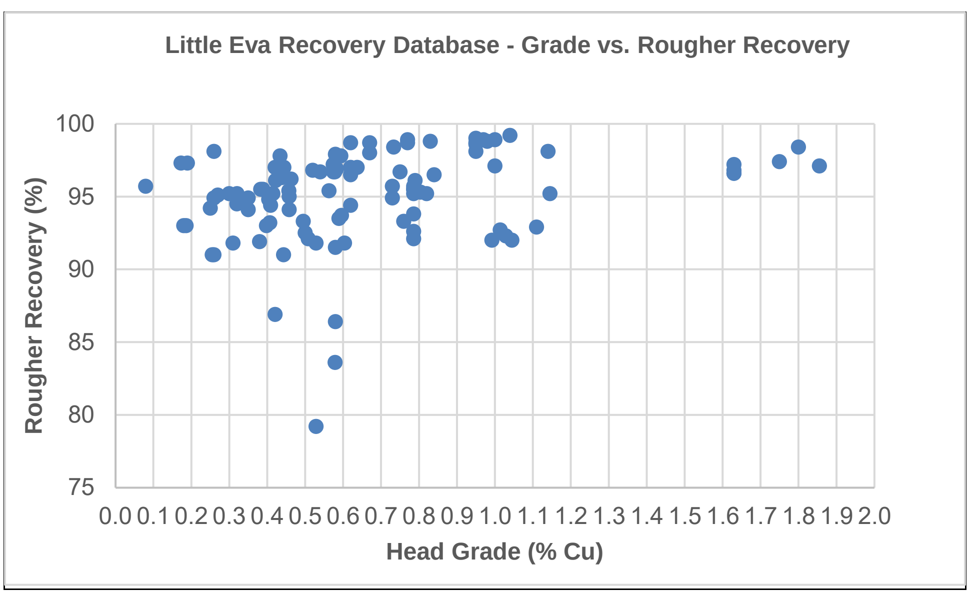


Figure 13-8: Little Eva Head Grade vs. Copper Recovery

The previous 2018 CMMC Feasibility Study was largely predicated on a readily available 32 ft x 18.75 ft SAG mill and 22 ft x 38 ft ball mill. The grind size of 250 µm was selected based on the economics of throughput versus grind and respective recovery. With the introduction of native copper material in a blended feed format, a finer grind is required. This updated feasibility study assumes a larger 24 ft x 40 ft ball mill, following a stage crushing circuit featuring an HPGR, which allows for a finer grind and higher throughput at a reduced operating cost.

The average rougher flotation recoveries from various test campaigns were sorted and compiled by grind to demonstrate the impact of liberation, as shown in Figure 13-9. At the new target grind of 165 µm, the target overall flotation recovery was set at 95% assuming 97% rougher recovery. Testwork completed by Ammtec (2012) provides additional detail on the flotation response at varying grind sizes for Little Eva master composites.

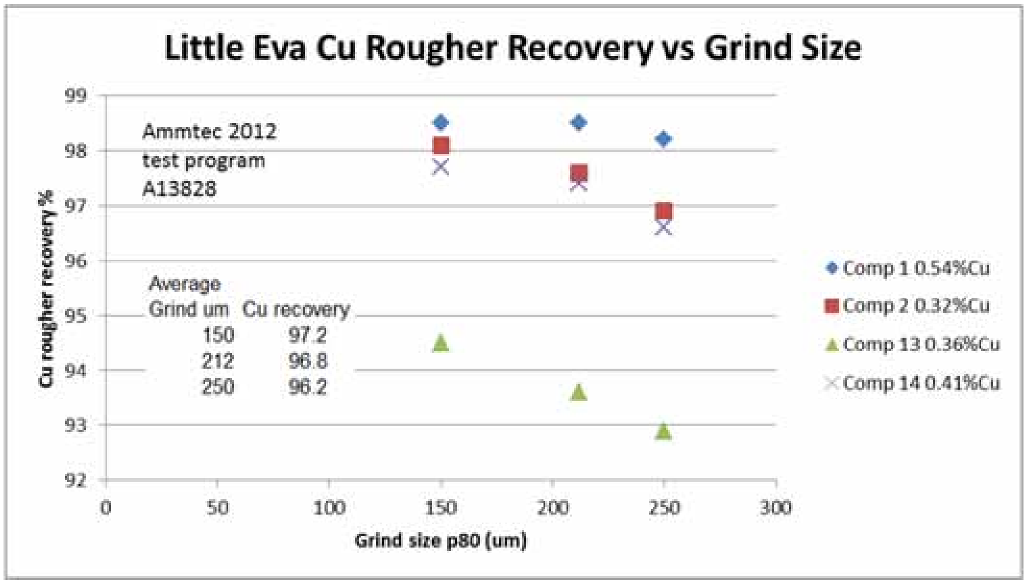


Figure 13-9: Little Eva Copper Rougher Copper Recovery vs. Grind Size P_{80} µm

A total of 51 cleaner stage tests have been completed throughout various studies, indicating saleable concentrate grades could be readily produced under most conditions. The data in the histogram below represents regrind product sizes varying from 38 µm to 75 µm, covering all results on file.

A design regrind size of 53 µm was selected based on the updated flowsheet design.

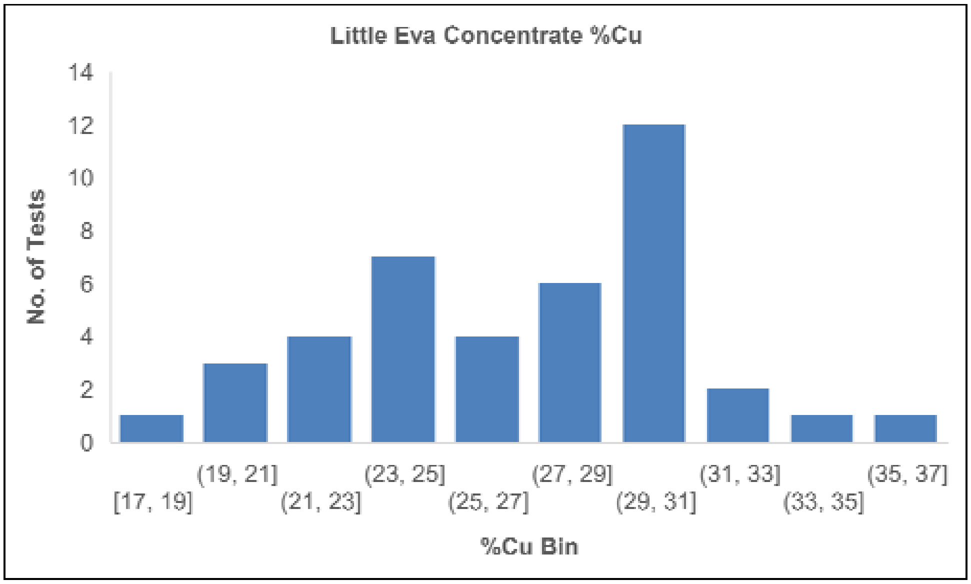
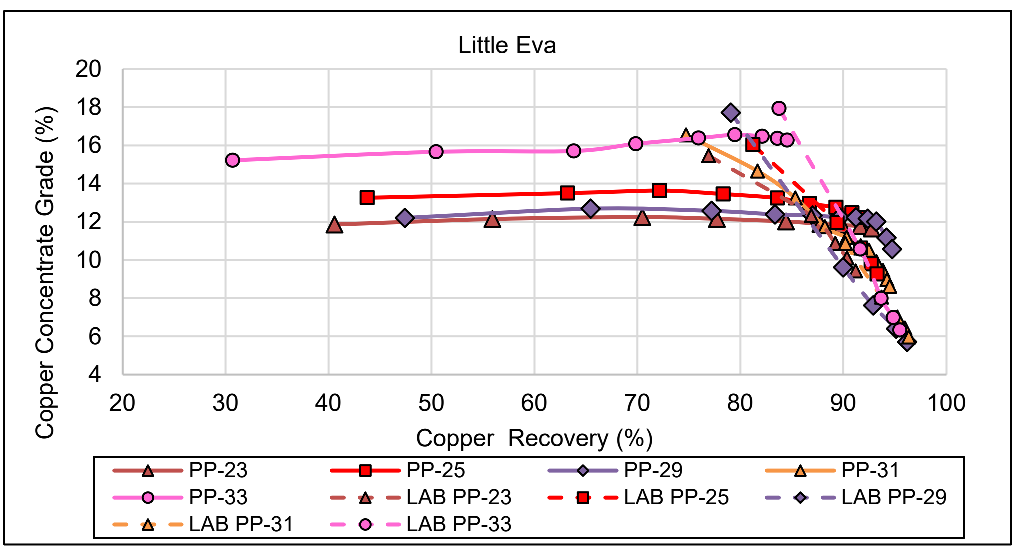


Figure 13-10: Little Eva Pit Concentrate Grade

13.2.4 Direct Flotation Reactors Program Results

The updated flowsheet also features two stages of cleaning using Woodgrove Direct Flotation Reactors (DFR) as the first and second cleaning stages. In April 2019, Little Eva pit samples were sent to the Copper Mountain Mine metallurgical laboratory to undergo pilot scale DFR cell testwork. The main approach was to validate rougher performance; however, cleaner performance of the Copper Mountain plant process rougher concentrate flows was also evaluated.

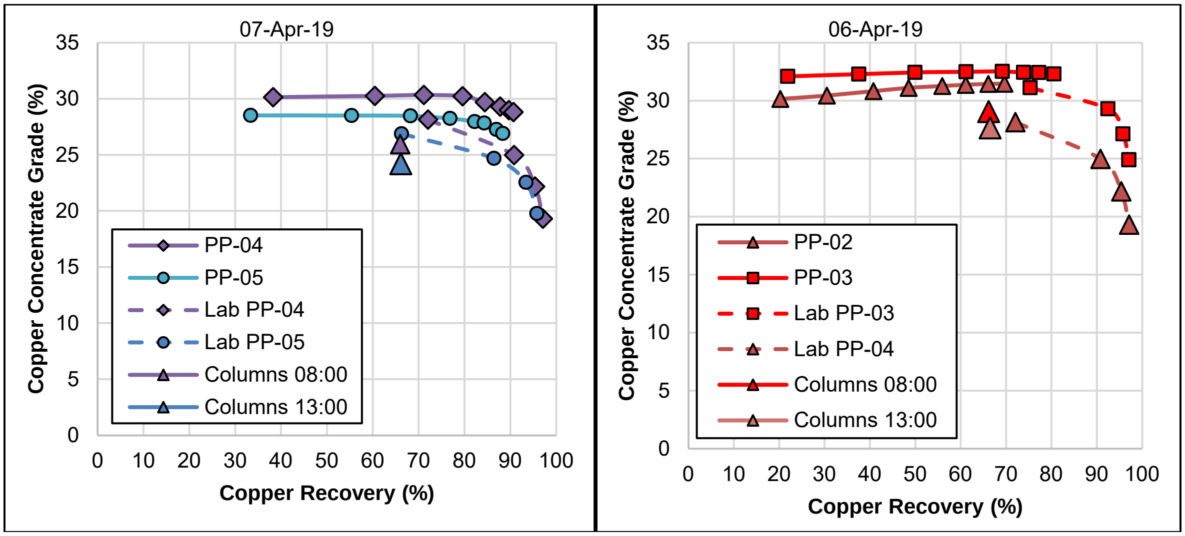
Figure 13-11 shows the DFR cell grade versus mass pull curves were able to achieve comparable rougher recovery at a higher grade when compared with a traditional benchtop mechanical cell.



Note: Solid lines represent DFR data, while dotted represents bench top cell results

Figure 13-11: Little Eva Ore DFR Test Results

As it was a challenge to generate enough mass to achieve a steady state for cleaner tests, in-process samples from the Copper Mountain Mine cleaner circuit were fed to the DFR pilot cell to determine a comparable performance versus the installed column cells. Results are shown in Figure 13-12.



Notes: Solid lines = DFR pilot cell data; dotted = bench data; Individual points = actual column performance at time of sampling.

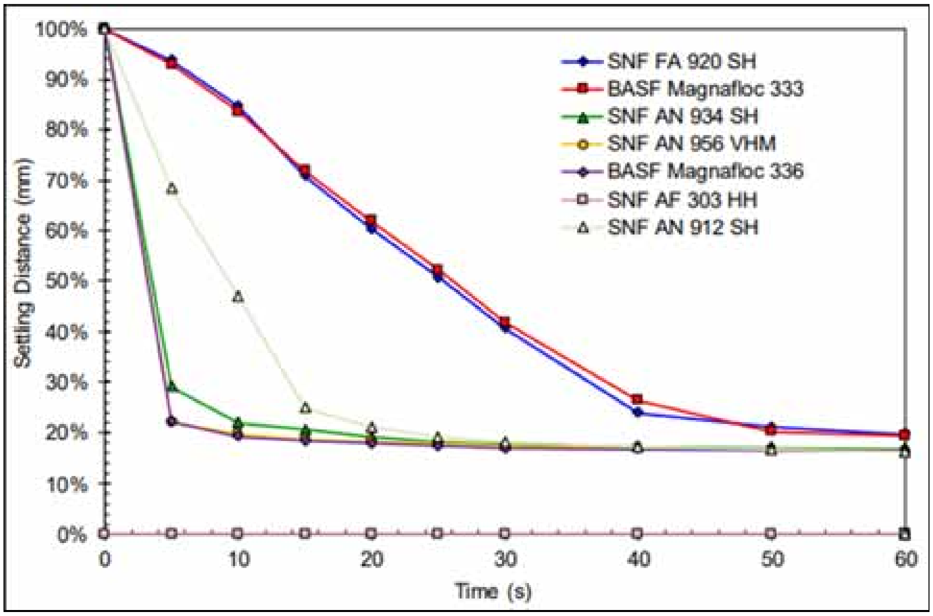
Figure 13-12: DFR Pilot Cell Results on In Process Copper Mountain Mine Cleaner Samples

Based on the above data, DFR cells were selected for cleaner circuit purposes; however, traditional cells were selected for rougher flotation, based on the size of the installation, and minimizing the risks associated with new technologies. The Copper Mountain Mine cleaner circuit data indicated a higher final concentrate grade could be achieved with the DFR cleaner cells versus the column cell used in the 2018 feasibility study. The cells proved to be more selective and efficient at gangue washing.

Based on the data shown above and in Figure 13-10, and the testwork performed, a final concentrate grade of 28% was selected for the design of the flotation circuit. This value is reasonable when compared with the historical locked-cycle and bench test programs; however, more confident given the application of the newer DFR technology.

13.2.5 Little Eva Tailings

Tailings generated from flotation tests performed by Copper Mountain Mine were sent to Patterson & Cooke in Denver for liquid-solids separation testing and thickener size determination. Little Eva tailings were tested, both individually and as a blend with Blackard material. Overall, the material performed well and supported a standard thickener loading rate of 1.0 t/h/m^2.



Source: Patterson & Cooke, 2019.

Figure 13-13: Little Eva Tailings Settling Profile

The above testwork was performed at a slightly coarser target grind of 180 µm. Several other characteristics were tested, with the results summarized in Table 13-2. The tailings sample was on average able to achieve a high underflow density with an acceptable amount of turbidity in the overflow.

Table 13-2: Little Eva Tailings Characterization Results

Property @ 180 µm Grind	Unit	Little Eva Tailings
Underflow Solids Density	%Solids w/w	71.6
Overflow Suspended Solids Mass Concentration	mg/L	252
Overflow Turbidity	NTU	397
Design Thickener Loading rate	$t/h/m^2$	1.00

Source: Patterson & Cooke, 2019.

The selected underflow density for the Project is 63%s w/w.

13.3 Sulphide Satellite Deposits

Within the Eva Copper Project land package, there are four satellite pits bearing high recovery sulphide ore types. In order of plant feed contribution these are: Turkey Creek, Bedford, Lady Clayre, and Ivy Ann. Turkey Creek, being the largest and representing 6.2% of the property reserves, is adjacent to the mill site and will provide high recovery ore to the mill for two years of production. The remaining pits are located south of the mill site and have varying copper grades and flotation performance. In total, 13 comminution tests and 53 flotation tests have been completed on the various sulphide satellite pits.

13.3.1 Sulphide Satellite Comminution

Historical testwork was validated with available core in 2019 for Turkey Creek, Bedford, and Ivy Ann pits with SMC and BWi work performed. In general, the pits demonstrated medium to high competency, and medium to high hardness. The values do not present a concern regarding the overall comminution circuit design, which is based on a blend of Little Eva and Blackard material.

Table 13-3: Sulphide Satellite Deposit Comminution Metrics (Average of Test Results)

Pit	Axb	BWi
Turkey Creek	31.7	13.2
Bedford	49.8	17.3
Lady Clayre	N/A	16.2
Ivy Ann	38.2	17.0

13.3.2 Sulphide Satellite Flotation

In general, the sulphide satellite pits contain copper present as chalcopyrite, except for Turkey Creek, which has large zones containing bornite and chalcocite.

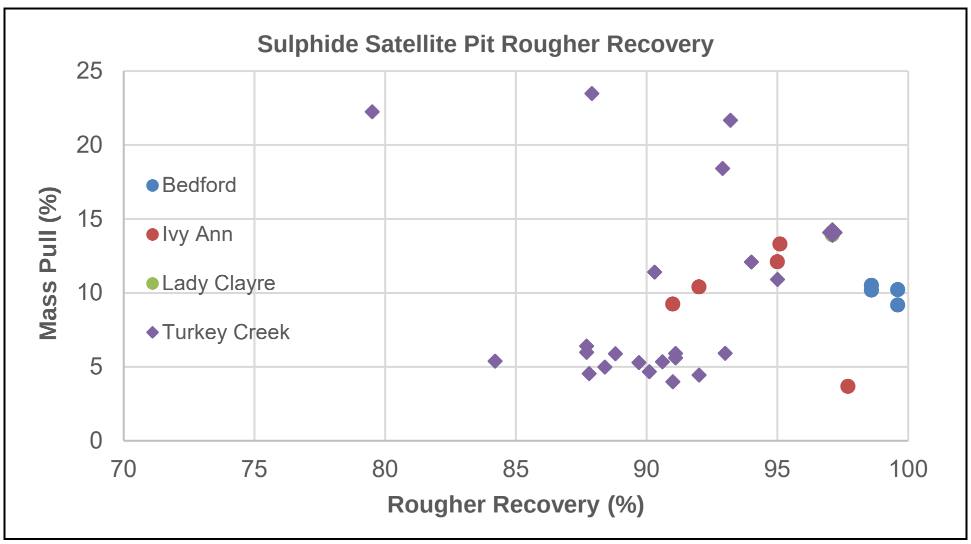
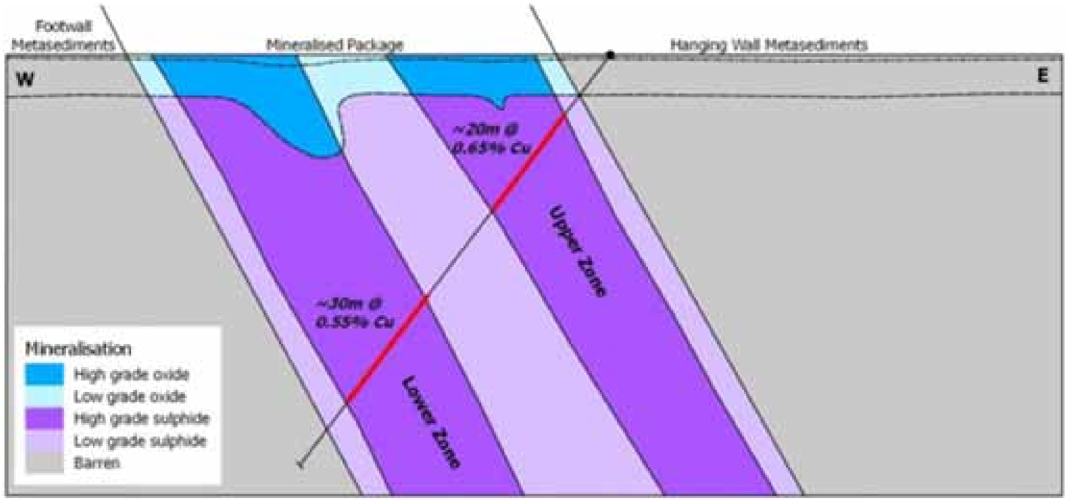


Figure 13-14: Sulphide Satellite Deposit Copper Recovery vs. Mass Pull

Turkey Creek demonstrates different mineralogy from the other sulphide satellite deposits in that copper is present as bornite and chalcocite in portions of the pit. Geochemical analysis has highlighted a distinct shift in mineralogy within parallel-trending layers referred to as the Upper and Lower zones (Figure 13-15). Metallurgical testwork was previously performed by Altona through ALS in 2015 and 2016, and was recently repeated by CMMC. The goal of the testwork was to validate whether there is a basis to qualify the upper and lower zones with distinct metallurgical performance.



Source: Altona, 2014.

Figure 13-15: Turkey Creek Pit Mineralization Zones

As shown in Table 13-4, in 2016, ALS prepared a bulk composite representing the full pit, and achieved 88% to 91% rougher copper recovery across various grind sizes. However, the bulk composite did not provide insight into whether the two banded zones were behaving differently on a recovery performance basis.

Table 13-4: Turkey Creek Bulk Composite Flotation Results (ALS, 2016)

Grind Size (P_{80} µm)	Product	Final Concentrate Date						
			Copper		Gold		Sulphur	
		Mass (%)	Grade (%)	Recovery (%)	Grade (%)	Recovery (%)	Grade (%)	Recovery (%)
212	RO Conc.	6.43	7.26	87.7	0.23	61.7	2.85	83.1
180		5.83	7.70	88.8	0.15	48.4	2.99	86.1
106		5.39	8.47	90.6	0.17	49.7	3.32	86.3
75		4.00	12.10	91.0	0.19	43.8	4.70	86.7

In 2019, samples from a fresh drill hole were sent to ALS to be composited in continuous lengths representing each of the upper and lower zones. Both zones demonstrated similar performance, again achieving 88% to 91% rougher recovery for nominal mass pulls and design head grade. Based on a project target grind of 165 µm, the overall copper recovery for this pit was set at 88%.

Samples from Bedford have been observed to contain large, high-grade, veiny grain structures, resulting in rougher copper recoveries as high as 99% with a mass pull of approximately 10%. With the current flotation circuit design, Bedford has been assigned a conservative overall recovery of 95%.

Historical Lady Clayre testwork indicated rougher copper recoveries varying from 84% to 97% with two locked-cycle tests achieving 94.5% overall recovery. A conservative estimate of 93% recovery has been set to account for the target grind.

Table 13-5: Updated Turkey Creek Flotation Performance – Grades and Recoveries (ALS, 2019)

Test No.	P_{80} (µm)	Wt (%)	Copper		Iron		Sulphur		Silver	
			%	% Recovery	%	% Recovery	%	% Recovery	g/t	% Recovery
Upper Total Core Comp. – RO Conc. 1-6										
MN2421	250	5.95	7.92	87.7	5.07	7.95	3.38	84.2	32	80.2
MN2420	180	5.89	8.17	91.1	5.22	8.15	3.48	84.5	33	80.4
MN2430	165	5.60	8.61	91.1	5.53	8.26	3.71	84.6	35	80.4
MN2419	106	5.92	8.49	93.0	5.51	8.55	3.64	88.4	34	81.3
Lower Total Core Comp. – RO Conc. 1-6										
MN2418	250	5.6	7.51	84.2	2.77	6.66	2.48	78.5	26	74.3
MN2417	180	5.07	8.14	89.7	3.05	6.87	2.78	83.2	28	75.3
MN2429	165	4.70	9.26	90.1	3.38	6.94	3.24	82.2	31	75.2
MN2416	106	4.41	9.94	92.0	3.68	6.99	3.46	88.9	34	76.6

Historical Ivy Ann testwork has typically yielded rougher copper recoveries between 91 and 95%; however, all previous testwork was done at a finer grind of 106 µm. Core from storage was pulled in 2019 and sent for confirmation testwork to ALS. Although this was a lower grade sample (0.21% Cu), it achieved a rougher copper recovery of 98% at a grind of 165 µm and nominal mass pull of 5.6%. This pit has been assigned a target overall recovery of 95%.

13.4 Native Copper Satellite Deposits

The native copper satellite pits, Blackard and Scanlan, can both be generally described as weathered copper deposits containing very soft rock, with copper deported as fine-grained native copper, intergrown with oxides overlying primary copper sulphides at depth. Blackard, being the larger contributor of this ore type, is a high-grade deposit containing 28% of the contained copper in the property ore reserve. This deposit, along with Scanlan, was the original focus of the Universal DFS (2009). However, at that time, Little Eva was a significantly smaller resource, resulting in a milling plan structured around processing native copper material with a smaller proportion of Little Eva sulphides blended into the mill feed. The present feasibility study includes use of native copper-bearing feeds as the minority component in a sulphide blend, at a ratio of 75:25 sulphide ore to native copper ore by feed mass.

Due to the variability and processing challenges associated with this ore type, 319 flotation tests have been performed on non-blended (no sulphide ore feed) ore samples from Blackard, and 16 on Scanlan. On a comminution basis, the majority of the samples have been shown to be very soft. The 70^{th} percentile of available hardness data is used for plant design.

Updated testwork was completed in 2019 to determine a suitable processing plan, which resulted in a flowsheet combining gravity and flotation circuits. The approach was based on successes and challenges seen at other operations with similar blends of ore types, such as Ernest Henry (previously) and New Afton (currently). In general, the approach was structured to address the following challenges:

- The material has demonstrated an average flotation recovery of 60.7%.
- Fine grained native copper is typically found in flotation tailings.
- A portion of the lost copper is present in the lattice and cleavage weathered mica (hydrobiotite). This portion is not amenable to flotation. If present in a high enough specific gravity (SG) relative to the host rock, this can be recovered in a gravity process.
- The ore typically demonstrates slow flotation kinetics. A longer residence time and late stage sulphidization of oxide-rimmed grains typically results in improved copper recovery in the rougher cells.

13.4.1 Native Copper Mineralogy

Multiple rounds of mineralogical analysis have been performed on Blackard and Scanlan material throughout prior studies. These reports discussed fine-grained native copper with lesser quantities of copper sulphides. A common discussion point was copper bound within hydrobiotite phases. This was described as both, copper bound within a biotite crystal structure, and interlaminar growth of very fine sheets.

A fresh sample generated from a bulk composite of two holes located in Blackard was sent for mineralogy at Process Mineralogical Consulting (PMC) in Maple Ridge, British Columbia, Canada. The sample was split into feed, gravity (Knelson) concentrate, and gravity (Knelson) tailings. The purpose was to achieve a better understanding of the overall mineralogy of the ore and an indication of the gravity-recoverable copper. The gravity (Knelson) concentrate stream was a single-pass concentrate generated from a typical Knelson gravity test.

Analysis of the feed sample yielded the following conclusions:

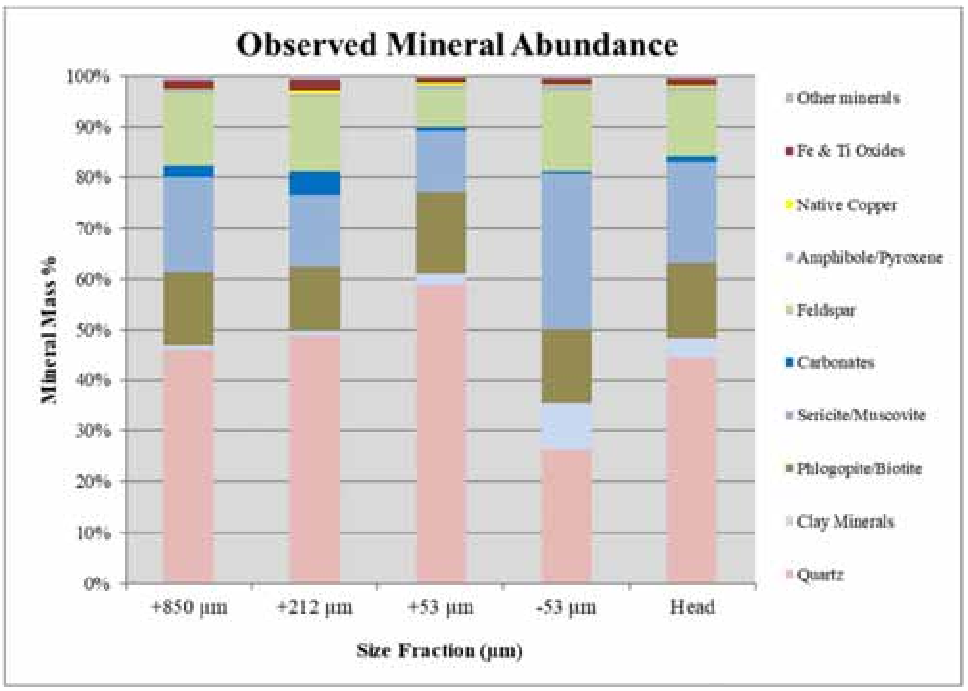
- 82.9% of total Cu was present as native copper. This was concentrated in the coarsest fractions (95.7% in +850 µm), and dropped off in the finer fractions (58.3% in +53 µm).
- 12% of Cu was present as sulphides, mainly chalcocite/covellite, with lesser quantities of bornite/chalcopyrite. The occurrence of sulphides increases in the finer fractions, as noted above.
- Cuprite was present as finely disseminated particles locked in gangue and intergrown with liberated native copper.
- Trace amounts of copper were detected as extremely fine-grained minerals bound within sheet silicates (mainly hydrobiotite).
- The average grain size of native copper particles was a P_{80} of 100 µm, whereas sulphides were observed at a P_{80} of 18 µm.
- On an SG basis, PMC noted that a theoretical maximum of 90% of copper could be recovered at a density split point of 4.0 to 6.0 grams per cubic centimeter (g/cm^3).

The ore is predominantly composed of quartz (43 %/w), feldspar (12.3 wt.%), muscovite/sericite (19.1 wt.%), biotite/phlogopite (14.3 wt.%) as well as minor amounts of clay minerals (kaolinite/dickite; 3.8 wt.%) and chlorite (2.1 wt.%). Other minor and accessory phases are mafic minerals such as amphiboles and pyroxenes (0.8 wt.%), carbonates (1.3 wt.%), and iron and titanium oxides/hydroxides (e.g., hematite, magnetite, 1.2 wt.%) and native copper (0.5 wt.%). Other minerals include chiefly epidote-group minerals and apatite.

Table 13-6: Blackard Feed Sample Copper Deportment

Minerals/Mass of Cu (%)	+850 µm	+212 µm	+53 µm	-53 µm	Head (Calc.)
Native Copper	95.6	91.2	87.4	58.2	82.9
Cuprite	2.17	4.15	4.77	4.89	4.24
Chalcopyrite/Bornite	0.11	0.48	1.84	3.00	1.54
Chalcocite/Covellite	1.70	3.52	5.10	32.5	10.5
Tetrahedrite/Tennantite	0.01	0.02	0.08	0.31	0.11
Cu Bearing Silicates	0.25	0.20	0.17	0.12	0.18
Cu Bearing Fe Oxyhydroxides	0.15	0.40	0.67	0.88	0.58
Total	**100**	**100**	**100**	**100**	**100**

Source: PMC, 2019



Source: PMC, 2019

Figure 13-16: Blackard Feed Mineral Abundance

Micrographs and analysis of mineral associations show complex particles of native copper, intergrown with chalcocite, cuprite, and gangue, as shown in Figure 13-17.

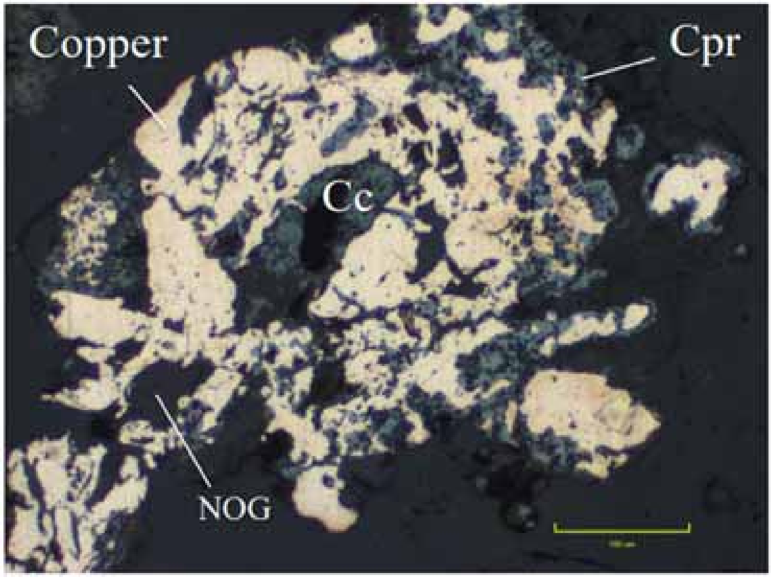


Figure 13-17: +212 µm Fraction Showing Native Copper, Chalcocite (Cc), and Cuprite (Cpr) (100 µm scale)

In the finer +53 µm fraction, increasing ratios of cuprite, chalcocite, and covellite are visible and present within complex particles.

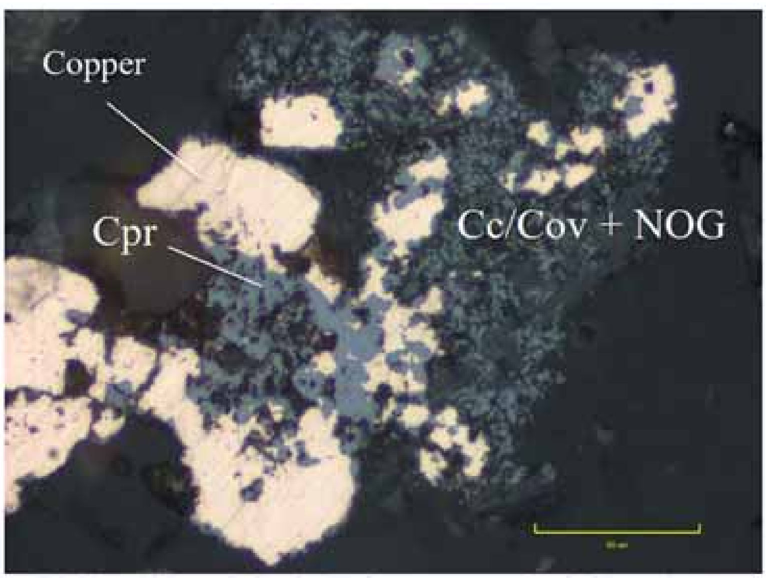
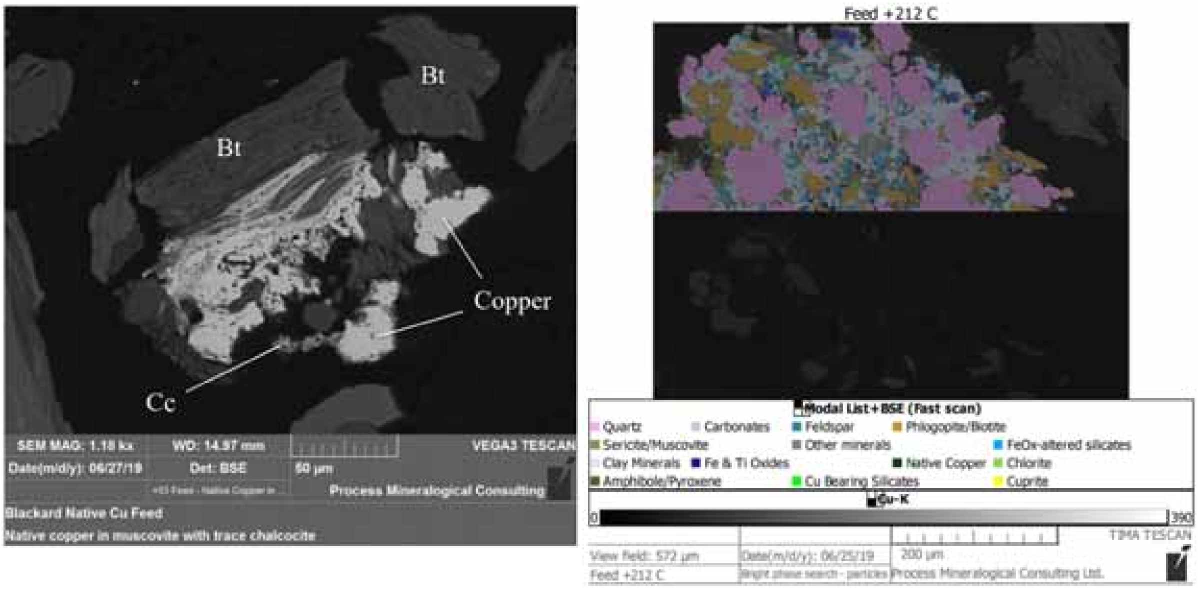


Figure 13-18: +53 µm Fraction Showing Native Copper, Chalcocite (Cc), Covellite (Cov), and Cuprite (Cpr) (100 µm scale)

Native copper associated with biotite was observed as being present as both inter-laminar sheets and as a solid solution within hydrobiotite (refer to Figure 13-19). This validated the results of previous studies, indicating that a portion of copper present is tied up within these hydrobiotite phases and is only likely to be recovered by means of SG differential within a gravity circuit.



Notes: The lower portion of the right-hand image was generated using Cu-Kα intensity mapping over biotite phases int the top-right frame (brown). (PMC, 2019)

Figure 13-19: Examples of Native Copper Associated with Sheet Silicates in the form of Both Inter-Laminar Growth (left), and Within Solid Solution of Biotite (right)

The gravity (Knelson) concentrate and gravity (Knelson) tailings streams showed the expected results, with a high deportment of coarse, native copper to the concentrate, and fine sulphides predominantly reporting to the tails.

Source: PMC, 2019

Figure 13-20: Gravity (Knelson) Concentrate Copper Modal Analysis

In summary, the mineralogical analysis provided the following conclusions to guide the selection of the copper recovery methods:

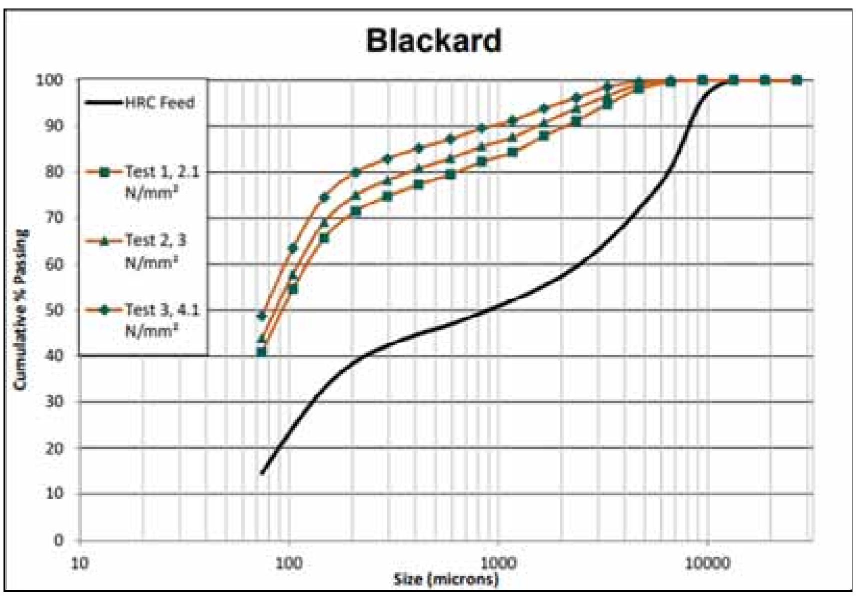
- The grain size and distribution of native copper should be amenable to gravity processing, such as a coarse jigging circuit.
- The gravity streams will recover a small portion of the sulphides present.
- A portion of the remaining sulphides and native copper in gravity tailings will be recovered with traditional flotation methods, with complex particles reporting to rougher concentrate.
- The deportment of chalcocite and bornite will require additional consideration in terms of flotation residence time and collector concentration and dosage.
- The presence of oxides in the finer fractions supports the use of sulphidizers to improve recovery.

13.4.2 Native Copper Ore Comminution

In total, 44 Blackard and 3 Scanlan comminution samples representing the native copper portions of each pit have been processed. On average, the samples exhibited very soft characteristics. The 70th percentile of the Blackard sample database showed an Axb of 149, and a BWi of 9.9. Likewise, the Scanlan samples 70th percentile showed an Axb of 66 and a BWi of 13.5.

A sample generated from the sulphide zone of Blackard was processed in 2019 and resulted in an Axb of 27 and a BWi of 14.8, more in line with other sulphide deposits on the property.

A sample of Blackard native copper material was also sent to the Metso York laboratory for HPGR performance characterization. The soft characteristics of the ore resulted in a higher quantity of fines following a single pass of the test HRC300; however, it resulted in a lower specific throughput. The results of this test informed the decision to set the Blackard blending rate at a ratio of 25% to 75 sulphide ore types to maintain higher production rates through the full scale HPGR.



Source: Metso, 2019

Figure 13-21: HRC300 Single Pass Test Results – Blackard Native Copper Zone

13.4.3 Native Copper Gravity Recovery Validation

Analysis of historical reports highlighted an opportunity to gain additional recovery with a larger focus on a gravity unit operation; however, this work previously struggled with attaining a high-grade final concentrate suitable for smelting. In 2019, this was revisited through working with equipment vendors that had seen success in similar sulphide/native copper blends at other operations. The opportunity is to capture traditional coarse free copper, and possibly additional mass present in sheet silicates and coarser particles, which are not amenable to sulphide flotation using traditional reagents. The secondary benefit of the gravity process is to remove native copper particles that typically concentrate to cyclone underflow of the ball mill and regrind mill.

Samples of core from the Blackard native copper zone were sent to Gekko Systems Pty Ltd (Gekko) located in Ballarat, Victoria, Australia, for determining the achievable copper recovery at a saleable concentrate grade. The intent was to validate a gravity recovery step located within the coarse ball mill cyclone feed loop prior to flotation.

A three-stage table gravity and magnetic separation upgrade process was used to determine the achievable copper recovery to a high-grade final concentrate. Two tests were completed on composites generated from two diamond drill cores. The first test yielded 13.4% Cu recovery at a final grade of 71.6% Cu, with the second run achieving a higher final grade of 87.5% Cu at a Cu recovery of 2.6%.

Table 13-7: Gekko Native Copper Gravity Results

	NC Test 1			NC Test 2		
	Stage Mass (%)	Stage Cu Grade (%)	Cu Distribution	Adjusted Mass (%)	Stage Cu Grade (%)	Cu Distribution
Table Feed	-	0.838	-	-	0.795	-
RGH Table Concentrate	3.22	7.9	30.3	3.09	6.53	25.4
Cleaner Table Concentrate	0.22	50.9	13.9	0.16	37.5	7.3
Non-Mags (Concentrate)	0.18	71.6	13.4	0.02	87.5	2.6
Gravity Tailings	**99.82**	**0.73**	**86.6**	**-**	**0.77**	**97.4**

Source: Gekko, 2019

A portion of the composite from the second test was then blended at a 25:75 ratio with Little Eva sulphide ore and sent to ALS labs in Perth to duplicate the process with the native copper material diluted with sulphide ore. Test results are shown in Table 13-8; this table also shows that the blended material continued to support gravity concentration to a saleable final concentrate grade.

Table 13-8: Native Copper and Sulphide Blended Gravity Run

	Blended Run		
	Stage Mass (%)	Stage Cu Grade (%)	Cu Distribution (%)
Table Feed	-	0.49	-
RGH Table Concentrate	12.30	1.7	42.5
Cleaner Table Concentrate	1.35	6.2	17.9
Non-Mags (Concentrate)	0.26	31.0	17.1
Gravity Tailings	-	0.41	82.9

Source: ALS, 2019

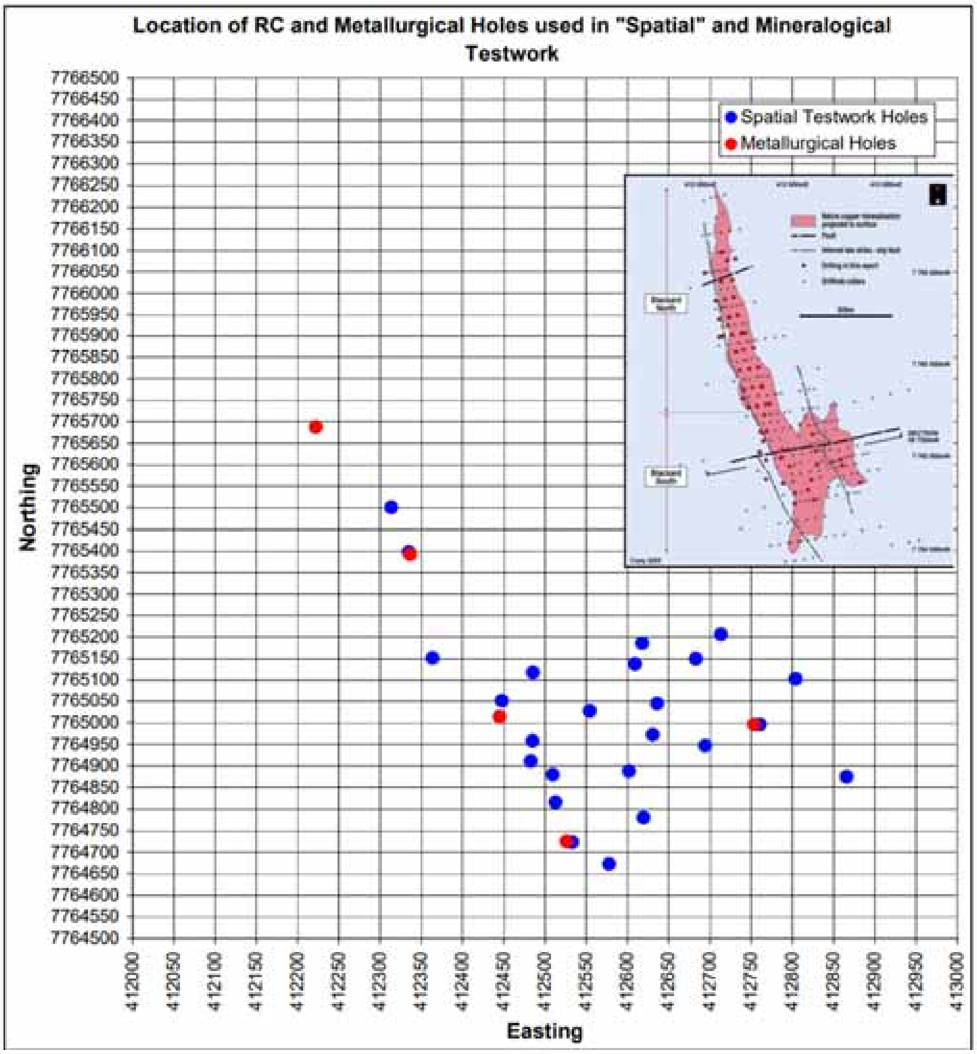
13.4.4 Native Copper Flotation Recovery

Blackard and Scanlan have been studied in multiple metallurgical campaigns ranging in scope from bench to pilot plant. In general, the studies indicated highly variable metallurgical performance depending on the location of the sample. The oxide cap results in poor flotation performance as expected. The native copper zone will achieve 50% to 70% Cu recovery, whereas the sulphide zone will achieve better than 90% Cu recovery to concentrate. It should be noted that in several historical reports, the term "oxide ore" refers to the native copper zone, rather than the true oxide cap of the deposit.

13.4.5 Native Copper Geometallurgical Bench Flotation Analysis

In 2006, Geostats Pty. Ltd. performed a statistical analysis of the Blackard deposit to determine the average metallurgical performance, using samples located spatially throughout the pit. A total of 29 samples were generated from five separate diamond core holes, which were partitioned by geological characteristics. These samples were floated on a rougher basis using a consistent method and reagent scheme. The goals of the analysis were to characterize the variability, and assign a target recovery to the native copper zone. Samples were located throughout the 2006 pit shell. The results of the analysis are shown in Figure 13-22. Average copper recoveries of 60.7% in rougher flotation and 2.3% in gravity concentration can be expected, resulting in an overall recovery of about 63%. The results of the analysis are shown in Table 13-9. Average copper recoveries of 60.7% in rougher flotation and 2.3% in gravity concentration can be expected, resulting in an overall recovery of about 63%.

The findings from the 29 metallurgical samples were later checked against results from 114 composites generated from samples pulled from 24 separate RC holes. Composites were generated in continuous lengths representing changes in geology, and grade ranges. The results were found to be in agreement with Figure 13-23.



Source: Geostats, 2006

Figure 13-22: Blackard Metallurgical Sample Sources for Spatial Analysis

Table 13-9: Bench Scale Blackard Native Copper Zone Flotation Results

Hole	Composite Description	Head Assays				Rougher Cu Recovery (%)
		Copper (%)	Gold (ppm)	Iron (%)	Sulphur (%)	
BCD452	Native Cu – Medium grade Cu	0.67	0.021	4.26	0.07	68.3
BCD452	Native Cu – High grade Cu	1.24	0.027	3.98	0.07	51.7
BCD452	Native Cu – Medium grade Cu	0.54	0.107	3.63	0.09	41.5
BCD452	Native Cu – Low grade Cu	0.3756	0.021	2.8	0.05	51.1
BCD453	Native Cu – Low grade Cu	0.4202	0.01	3.94	0.03	64.8
BCD453	Native Cu – Medium grade Cu	0.83	0.01	4.22	0.03	51.9
BCD453	Native Cu – Low grade Cu	0.3593	0.031	3.36	0.05	55.8
BCD482	Native Cu – Low grade Cu	0.2173	0.01	4.08	0.03	89.4
BCD482	Native Cu – Medium grade Cu	0.72	0.01	4.19	0.04	60.5
BCD482	Native Cu – Low grade Cu	0.3174	0.01	4.82	0.05	60.7
BCD482	Native Cu – Medium grade Cu	0.52	0.01	4.04	0.04	67.8
BCD482	Native Cu – Low grade Cu	0.303	0.01	3.49	0.04	69.3
BCD482	Native Cu – Medium grade Cu	0.71	0.01	2.71	0.04	59.9
BCD482	Native Cu – High grade Cu	0.92	0.056	4.57	0.05	50.5
BCD483	Native Cu – High grade Cu	0.92	0.022	3.52	0.05	62.8
BCD483	Native Cu – Low grade Cu	0.53	0.021	4.82	0.04	68.3
BCD483	Native Cu - Medium grade Cu	0.61	0.01	3.75	0.05	60.5
BCD483	Native Cu – Low grade Cu	0.4998	0.01	4.47	0.06	73.5
BCD483	Native Cu – Low grade Cu	0.2445	0.01	3.92	0.04	49.7
BCD484	Native Cu – High grade Cu	0.88	0.01	3.8	0.04	66.8
BCD484	Native Cu – High grade Cu	0.95	0.01	3.86	0.04	64.7
BCD484	Native Cu – High grade Cu	0.84	0.01	4.3	0.05	62.3
BCD484	Native Cu – Low grade Cu	0.4494	0.01	3.28	0.05	56.4
BCD484	Native Cu – High grade Cu	0.78	0.01	4.6	0.04	49.3
					Average	**60.7**

Source: Geostats, 2006

Blackard Oxide Copper Recovery

Native copper oxide zone "flotation only" Copper Recovery estimate: **60.74%**

Adjustment for Gravity Recovery: **+ 2.28%**

Native copper oxide zone Total Copper Recovery estimate: **63.02%**

Figure 13-23: Key Findings of 2006 Geostats Report

13.4.6 Native Copper Locked-Cycle Testing

In 2006, a series of locked-cycle tests were performed on Blackard material. A master composite labelled "Master Oxide" was compiled from diamond drill core and rejects from the above mentioned Geostats metallurgical program. Cuts of the composite were sent to Ammtec in Perth, and AMML in Gosford, for parallel analysis.

On the same composite, initial results yielded 45% Cu recovery, subsequently improved to 61% as conditions were optimized. No coarse gravity concentrates were generated; however, a panned concentrate at the cleaner stage was included in latter runs.

The key takeaways from the testwork can be summarized as follows:

- Aggressive collector dosage is required.
- The cleaner circuit generated a high-grade final concentrate. Pulling additional mass and driving the final concentrate grade down yielded higher cleaner performance.
- A gravity stage was required in the cleaner stage to mitigate losses. It was believed that most of the cleaner losses were due to liberated native copper escaping to the tailings stream.

Universal had taken these results and performed mass balancing on the data using LIMN (flowsheet simulator). The conclusion was that, following corrections, the balanced overall copper recovery for the oxide (native copper) composite was 64% (NeoProTec, 2008).

Table 13-10: 2006 Locked Cycle Test Results

	Master Oxide					Master Blend 1			Master Blend 2			Master Sulphide 2	
Lab	AMMTEC	AMML	AMML	AMML	AMML	AMMTEC	AMML	AMML	AMMTEC	AMML	AMML	AMML	AMML
Test No	GS3079	T13	T14	T36	T37	GS3085	T7	T22	GS3092	T15	T21	T6	T23
Actual Head	0.637	0.630	0.630	0.630	0.630	0.681	0.681	0.681	0.700	0.700	0.700	0.786	0.786
Calc Head	0.543	0.651	0.637	0.690	0.655	0.584	0.594	0.635	0.643	0.669	0.638	0.695	0.720
Grind 80% passing size µm	106	106	106	106	106	180	180	180	180	180	180	180	180
Tail Cu %	0.30	0.28	0.26	0.27	0.26	0.14	0.16	0.16	0.16	0.16	0.16	0.03	0.06
Tail wt %	99.3	96.6	96.3	98.8	98.3	98.4	97.8	97.2	97.2	96.2	96.4	97.6	97.3
Tail wt g	986.0	954.7	948.5	962.5	962.6	979.8	964.6	957.0	959.1	945.1	938.5	967.0	959.5
Total Product Cu Grade %	38.0	11.1	10.3	34.7	23.5	27.1	20.1	17.0	17.6	13.6	13.4	28.1	24.3
Total Product Cu Recov %	45.9	58.5	60.2	60.7	60.6	76.4	73.4	75.8	75.9	77.0	75.7	96.4	91.8
Total Product Wt %	0.65	3.42	3.71	1.21	1.69	1.65	2.17	2.83	2.78	3.78	3.62	2.38	2.72
Total Product Wt g	6.5	33.8	36.6	11.8	16.5	16.1	21.4	27.8	27.4	37.2	35.2	23.6	26.9
ReClnr Conc 1 Cu Grade %	52.1	14.3	5.5	***73.8***	***52.4***	37.0	41.2	32.0	32.5	21.4	20.4	31.6	28.5
ReClnr Conc 1 Cu Rec %	29.0	20.8	7.6	***45.2***	***52.4***	52.8	45.7	36.1	44.2	46.9	40.9	65.4	69.8
ReClnr Conc 1 Wt %	0.30	0.95	0.89	***0.42***	***0.54***	0.83	0.66	0.72	0.88	1.47	1.28	1.44	1.76
ReClnr Conc 1 Wt g	3.0	9.4	8.8	***4.1***	***5.29***	8.2	6.5	7.8	8.7	14.4	12.5	14.3	17.4
ReClnr Conc 1 to 2 Cu Grade %	45.1	9.6	4.2	***65.3***	***44.6***	31.6	40.4	25.6	26.3	16.3	15.7	30.8	26.6
ReClnr Conc 1 to 2 Cu Rec %	42.4	26.2	11.7	***51.9***	***52.0***	71.3	65.5	50.2	56.0	56.1	50.7	92.6	86.8
ReClnr Conc 1 to 2 Wt %	0.51	1.76	1.77	***0.55***	***0.76***	1.32	0.96	1.24	1.38	2.31	2.07	2.09	2.35
ReClnr Conc 1 to 2 Wt g	5.1	17.6	17.4	***5.4***	***7.44***	12.9	9.5	12.3	13.6	22.7	20.1	20.7	23.2
ReClnr Conc 1 to 3 Cu Grade %	38.0	7.4	3.4	***54.9***	***36.7***	27.1	39.3	21.2	22.1	13.0	11.8	28.1	24.3
ReClnr Conc 1 to 3 Cu Rec %	45.9	29.3	14.6	***56.0***	***56.3***	76.4	69.4	55.5	58.3	59.6	54.1	96.4	91.8
ReClnr Conc 1 to 3 Wt %	0.7	2.59	2.73	***0.70***	***1.00***	1.65	1.05	1.66	1.70	3.08	2.93	2.38	2.72
ReClnr Conc 1 to 3 Wt g	6.50	25.7	26.9	***6.8***	***9.79***	16.1	10.4	16.4	16.8	30.3	28.5	23.6	26.9
ReClnr Conc 1 to 4 Cu Grade %	-	-	-	***48.6***	***32.5***								
ReClnr Conc 1 to 4 Cu Rec %	-	-	-	***57.2***	***58.0***								
ReClnr Conc 1 to 4 Wt %	-	-	-	***0.81***	***1.17***								
ReClnr Conc 1 to 4 Wt g	-	-	-	***7.89***	***11.46***								
ReClnr Conc 1 to 5 Cu Grade %	-	-	-	***42.1***	***29.9***								
ReClnr Conc 1 to 5 Cu Rec %	-	-	-	***59.1***	***58.8***								
ReClnr Conc 1 to 5 Wt %	-	-	-	***0.97***	***1.29***								
ReClnr Conc 1 to 5 Wt g	-	-	-	***9.45***	***12.63***								
ReClnr Conc 1 to 6 Cu Grade %	-	-	-	***34.7***	***23.5***								
ReClnr Conc 1 to 6 Cu Rec %	-	-	-	***60.7***	***60.6***								
ReClnr Conc 1 to 6 Wt %	-	-	-	***1.21***	***1.69***								
ReClnr Conc 1 to 6 Wt g	-	-	-	***11.8***	***16.55***								
Cu % Grade to Grind Coarse Screen	-	11.1	15.2	-		-	2.1	2.5	5.0	4.0	5.4	-	-
Cu Rec to Grind Coarse Screen	-	11.2	15.6	-		-	4.0	4.0	4.4	3.3	4.4	-	-
Grind Coarse Screen Wt %	-	0.66	0.65	-			1.12	1.01	0.57	0.56	0.53	-	-
Grind Coarse Screen Wt g	-	6.5	6.5	-			11.0	10.0	5.7	5.5	5.1	-	-
Cu % Grade to Clnr Feed gravity	-	-	58.1	90.3	84.7	-	-	68.0	17.0	66.4	67.9	-	-
Cu Rec to Clnr Feed gravity	-	-	29.9	22.2	22.5	-	-	16.3	13.2	14.0	17.2	-	-
Clnr Feed gravity Wt %	-	-	0.33	0.17	0.17	-	-	0.15	0.50	0.14	0.16	-	-
Clnr Feed gravity Wt g	-	-	3.3	1.66	1.66	-	-	1.5	5.0	1.4	1.6	-	-
Cu % Grade to Clnr Feed screen	-	70.0	-	-		-	-	-	-	-	-	-	-
Cu Rec to Clnr Feed screen	-	18.0	-	-		-	-	-	-	-	-	-	-
Clnr Feed screen Wt %	-	0.2	-	-		-	-	-	-	-	-	-	-
Clnr Feed screen Wt g	-	1.7	-	-		-	-	-	-	-	-	-	-
Total g/t PAX	175	220	220	178	183	175	175	190	195	195	195	175	0
Total g/t A3477	70	80	80	80	80	70	80	80	90	80	80	80	8
Total g/t NaHS	90	110	110	100	110	90	90	105	90	105	105	90	0
Total g/t MIBC	?	90	90	65	70	?	180	100	?	100	100	210	60

Source: NeoProTec, 2006
Note: In tests T36 & T37, the values highlighted and underlined includes the gravity components.

13.4.7 Native Copper Pilot Plant Tests

In 2006, bulk samples from Blackard, Scanlan, and Little Eva were sent to Ammtec for pilot plant testing. At that time, the Little Eva resource had not been fully developed, resulting in native copper being the larger ore component considered in the plant design. Pilot tests were thus performed with a much higher percentage of native copper material (62.5% by mass).

Bench tests performed on the components indicated a target recovery of 75% when adjusted for the component recoveries and grades. Test 1 achieved this, whereas Test 2 suffered losses in both the rougher and cleaner stages. The reasons given for the lower performance were sampling errors and insufficient residence time in the cleaner circuit. The LIMN balanced pilot plant performance was lower than bench validation work on the same ore blend.

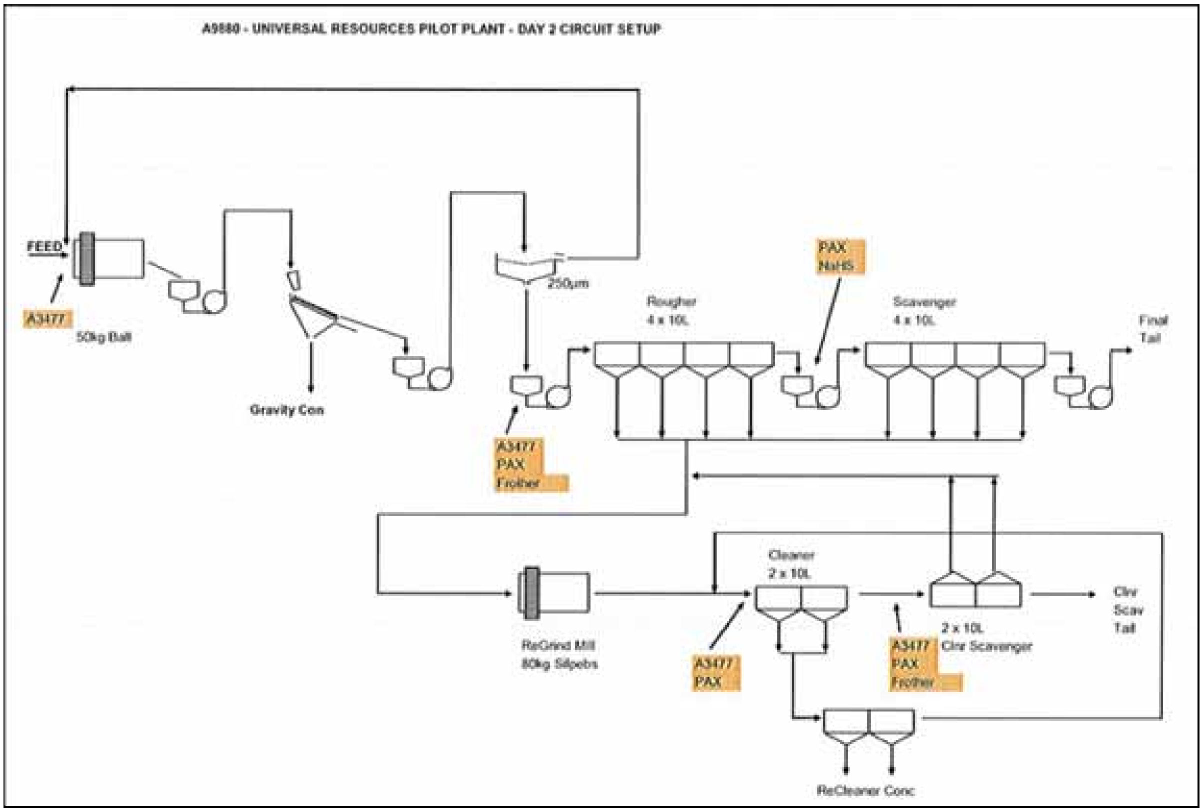
Test 3, using the Scanlan composite, showed higher than expected results; however, it was later reported that two holes were used for this composite, SCD 134 and SCD 135. These holes had bench scale copper recoveries of 55% and 75% respectively, indicating that the component recovery of this native copper composite was likely higher than the target of ~60%.

Table 13-11: Native Copper and Sulphide Blend Pilot Plant Results

Pilot Plant Run Number	1	1	3
Oxide Feed Blend Content	62.5% Blackard	62.5% Blackard	62.5% Scanlan
Sulphide Feed Blend Content	37.5% Little Eva	37.5% Little Eva	37.5% Little Eva
Rougher Recovery	76.8%	68.2%	83.3%
Recleaner Concentrate			
Copper Grade	24.5%	29%	31%
Copper Recovery	74.5%	64.7%	78.6%

Source: NeoProTec, 2006

The notes from this campaign highlighted challenges with overloading the Strake table used to simulate a coarse gravity concentrate, suggesting that there is an opportunity to improve this unit operation, as discussed in the above sections. It can be seen in Figure 13-24 that no gravity stage was present in the cleaner circuit, which contributed to cleaner stage losses that were subsequently recycled to the roughers and lost to final tails. As shown in the above locked-cycle test, this is another opportunity to extract free native copper within the cleaner circuit that is not amenable to flotation.



Source: NeoProTec, 2006

Figure 13-24: Pilot Plant Flowsheet

13.4.8 Blackard Sulphide Performance

Below the native copper zone within the Blackard pit is a sulphide zone. A composite from this zone was sent for mineralogy and flotation testwork in 2019 to determine performance. Copper was found to be primarily present as 69.9% bornite and 25.4% chalcopyrite. Only trace amounts of native copper were found.

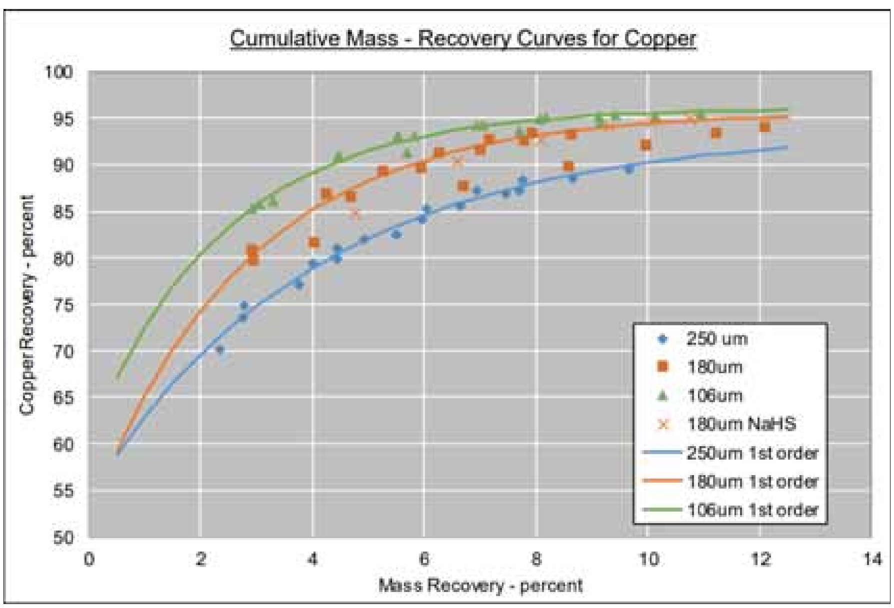
Table 13-12: Blackard Sulphide Zone Mineral Distribution

Sample Fraction	+212 µm	+106 µm	-106 µm	Head
Mass%	68.3	7.52	24.1	100
Chalcopyrite	0.27	0.24	0.23	0.26
Bornite	0.36	0.63	0.73	0.47
Chalcocite/Covellite	0.02	0.04	0.01	0.02
Tetrahedrite/Tennantite	0.03	0.02	0.02	0.03
Native Copper	0.00	0.00	0	0.00
Pyrite-Pyrrhotite	0.00	0.00	0	0.00
Apatite	0.30	0.25	0.37	0.32

Sample Fraction	+212 µm	+106 µm	-106 µm	Head
Calcite	30.03	31.2	29.6	30.2
Fe & Ti Oxides	0.23	0.24	0.40	0.27
Quartz	29.2	30.5	33.3	30.3
Feldspar	11.8	9.98	9.82	11.2
K-Feldspar	5.74	6.45	5.25	5.67
Phlogopite/Biotite	9.05	8.62	8.49	8.89
Sericite/Muscovite	6.25	5.94	5.65	6.08
Epidote	1.10	0.88	0.89	1.03
Amphibole/Pyroxene	3.04	2.50	2.35	2.83
Chlorite	2.01	2.35	2.56	2.17
Clay Minerals	0.14	0.12	0.08	0.13
Other Minerals	0.08	0.06	0.21	0.11
Total	**100**	**100**	**100**	**100**

Source: PMC, 2019

A cut from the same composite was sent to Base Metallurgical Laboratories (Basemet) in Kamloops, British Columbia, to determine flotation performance. Copper recovery in rougher flotation was shown to vary between 89% to 95% at a mass pull of 10%.



Source: Basemet, 2019

Figure 13-25: Blackard Sulphide Rougher Mass Recovery Curves

13.5 Blended Ore

The plant design is based on a blend of sulphide ore and native copper ore, not exceeding 25% of native copper. Design inputs were generated based on this ratio, and the individual component metrics.

13.5.1 Blended Ore Comminution

The average comminution parameter inputs were based on the 70th percentile of the Little Eva and Blackard databases to ensure high 'installed power' confidence.

Table 13-13: Design Comminution Inputs

Parameter	Unit	Sulphide (Range)	Native Copper (Range)	Design Value (70th Percentile)
Bond Crushing Work Index	kWh/t	3–23	1–14	14
Axb	-	31–204	48–998	50
Bond Rod Mill Work Index	kWh/t	11–24	4–15	18.7
Bond Ball Mill Work Index	kWh/t	6–22	3–16	16.5
Bond Abrasion Index	kWh/t	0.03–0.53	0.01–0.02	0.17
HPGR Specific Throughput	ts/m^{3}h	311	230	291

The result of the parameters above is a design grind of 165 µm when processing the ore blend at 31,200 t/d.

13.5.2 Blended Ore Gravity Recovery

As shown in Table 13-14, a saleable final concentrate grade can be achieved from the blended ore material when it is processed through a jig. This was demonstrated in testing by tabling; however, in a full-scale application, native copper will naturally concentrate to cyclone underflow, which will further assist in gravity concentrating to a higher grade.

The design of this portion of the mill is based on the success of similar operations, such as the New Afton mine in Canada. New Afton is successfully utilizing the same jigs specified for this Project to extract a high-grade gravity concentrate while processing a similar blend at a ratio as high as 30% native copper. The format and design for the Eva gravity recovery flowsheet was based on successes and challenges noted by operational staff at New Afton mine.

A gravity bowl concentrator has been included in the regrind cyclone loop to address the concern of free native copper being lost in cleaner flotation, as highlighted in the above analysis of the locked-cycle test results.

13.5.3 Blended Ore Flotation

All testwork on sulphide ore types have yielded strong performance and fast kinetics. Sulphidization has typically resulted in an increase of kinetics of native copper ore types in the last stages of flotation, indicating recovery of particles rimmed with cuprite and other oxides. The roughers will have provision for addition of sulphidizer in the last two stages of flotation.

Table 13-14: Blended Ore Flotation Design Inputs

Parameter	Unit	Design Value
Target Primary Grind	P_{80} (µm)	165
Rougher Cu Recovery	%	91.4
Rougher Mass Pull	%	10
Target Regrind	P_{80} (µm)	53
Cleaner Cu Recovery	%	98
Target Final Concentrate Grade	% Cu	28

13.5.4 Concentrate Analysis

Analysis of concentrates has been performed through various flotation campaigns and the 2006 pilot plant test. A further check was performed on the high-grade gravity concentrate generated in the Gekko gravity testwork on Blackard material. In all cases, no penalty elements were noted at levels that could cause concerns for potential treatment facilities.

Table 13-15: Concentrate Analysis

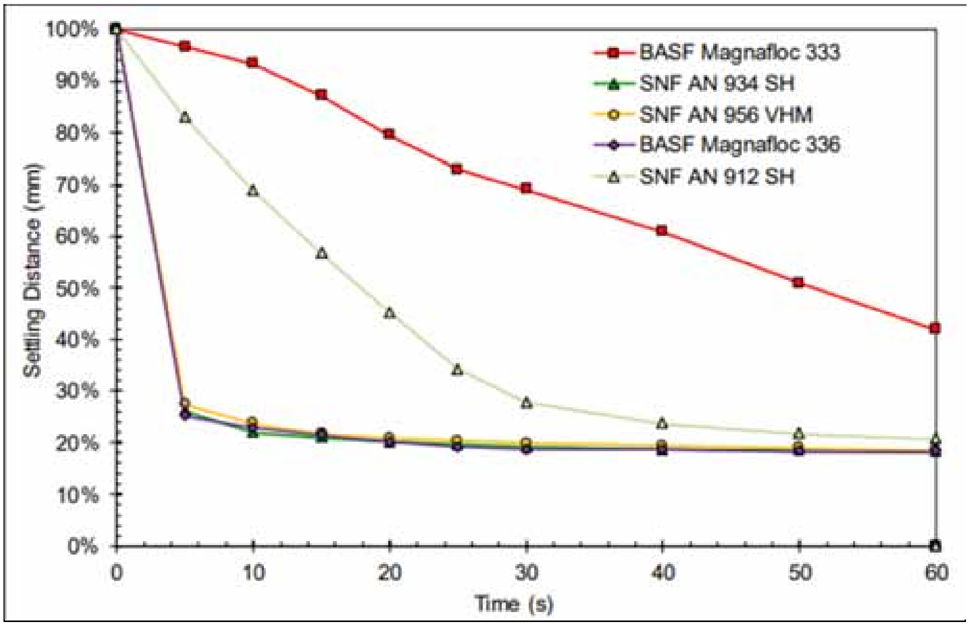
Element	Unit	Eva Float Comp. 1 GS6060	Eva Float Comp. 13GS6123	2006 Pilot Plant Blend (Eva & Blackard)	2019 Blackard Gravity Concentrate
Ag	ppm	12	44	20	14.9
Al	%	1.84	2.04	0.86	0.45
As	ppm	<10	20	21	40.6
Au	ppm	3.11	3.9	4.52	15.5
Ba	ppm	180	40	4.41	10
Be	ppm	0.1	<5	102	1.95
Bi	ppm	<10	70	<10	22.2
Ca	%	1.6	0.9	0.33	0.39
Cd	ppm	10	15	<5	0.23
Cl	ppm	ins	ins	<50	0
Co	ppm	20	10	30	13.3
Cr	ppm	160	400	161	7
Cu	%	24.4	28.7	33.8	64.6
F	ppm	170	ins	130	0
Fe	%	23.1	18.5	22	4.22
Hg	ppm	0.2	ins	0.3	0.16
K	%	0.57	0.8	0.18	0.14
Li	ppm	10	<5	<5	6.9
Mg	%	0.56	0.32	0.19	0.32
Mn	ppm	<100	ins	259	0.57

Element	Unit	Eva Float Comp. 1 GS6060	Eva Float Comp. 13GS6123	2006 Pilot Plant Blend (Eva & Blackard)	2019 Blackard Gravity Concentrate
Mo	ppm	420	575	688	0.65
Na	%	0.99	0.89	0.38	0.01
Ni	ppm	145	265	281	11.4
P	ppm	300	<100	266	530
Pb	ppm	100	355	212	34.2
S	%	25	23.7	26.4	0.17
Sb	ppm	0.7	ins	82	0.01
SiO_2	%	11.4	15.6	6.25	14.85
Sn	ppm	<50	<50	2.3	0.5
Sr	ppm	20	50	27	17.4
Te	ppm	3.2	ins	0.77	0.43
Th	ppm	4	2	1502	6.1
Ti	ppm	2,600	0.18	3.5	0.024
U	ppm	4.8	2.1	<50	3.41
V	ppm	58	ins	38	31
Y	ppm	10	10	19	17.25
Zn	ppm	1,236	36	173	20
Zr	ppm	8	55	60	5.5

Note: ins = insufficient sample for analysis

13.5.5 Blended Ore Tailings

Tailings from Little Eva and Blackard flotation tests were sent to Patterson & Cooke in Denver, Colorado, USA, for tailings characterization. The tailings samples were blended to match the 75% sulphides 25% native copper ratio and tested for high-rate thickening efficiency. The results indicated no detrimental effects on liquid-solids separation resulting from blending the ore types, with corresponding thickener underflow densities of 62.5% to 65.8% by weight.



Source: Patterson & Cooke, 2019

Figure 13-26: Eva and Blackard Tailings Blend Settling Profile

13.6 Metallurgical Conclusions

The sulphide portion of the resource, comprising the Little Eva, Turkey Creek, Bedford, Lady Clayre, and Ivy Ann pits, are relatively straightforward from a processing perspective. Little Eva, being the bulk of this ore volume, has routinely shown high flotation performance with coarse mineralization and fast kinetics.

Some of the satellite pits can present unique challenges in terms of a shift in deportment from chalcopyrite to bornite and chalcocite; however, this remains within the scope of the plant design, with expected recoveries adjusted accordingly. The sulphide portion located below the native copper zone of the native copper pits performs similarly to Turkey Creek and will demonstrate high copper recovery.

Testwork results for native copper pits (Blackard and Scanlan), have demonstrated highly variable metallurgical recoveries. A combined flowsheet including gravity and flotation unit operations is required to address the changing metallurgy performance as these pits are mined. The geospatial work performed on Blackard ore by Geostats provides confidence in terms of average metallurgical performance; however, variability between subsequent phases of mining is to be expected. Metallurgical data on the Scanlan pit is sparse relative to other portions of the Project; however, pilot work and preliminary bench flotation work performed by Universal prior to 2009 provides indications that Blackard and Scanlan share similar geological and metallurgical performance characteristics. Geologically, Scanlan is considered the same as Blackard, and the metallurgical data supports this; however, there is a risk associated with the low amount of testwork data to date.

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

The Eva Copper Project is currently composed of seven deposits; in order of importance, they are Little Eva, Turkey Creek, Blackard, Scanlan, Bedford, Ivy Ann, and Lady Clayre. Little Eva is the main deposit, hosting a majority of the Mineral Resource and Reserve, while the others are considered satellite or supplemental deposits. As there are significant differences in the deposits with respect to tonnage, metal grades, nature of mineralization, and drill density, different resource estimation strategies were employed for each deposit. The geology, structural setting, and mineralization of each of the deposits has been described in previous sections and will only be touched upon in this section as is required for understanding resource estimation.

All deposits have had previous resource and reserve estimates carried out. Some additional drilling has been carried out on the Little Eva, Turkey Creek, and Blackard deposits since the previous resource estimates were made, but the amount of drilling, in comparison to past work, was relatively minor, as the new drilling was mostly for verification of historical data and to collect material for metallurgical testing. For the most part, resources have been re-estimated using different techniques and block sizes to better match proposed mining equipment and incorporate anticipated mining dilution and ore losses associated with the larger equipment. The type of mineralization is such that the larger mining equipment, while increasing mining efficiency, will likely result in higher levels of dilution with resultant lower grades. However, the amount of contained metal within the earlier and current estimates is similar. It is anticipated that there will be opportunities to increase grades delivered to the mill through enhanced grade control procedures during mining together with the use of stockpiling strategies. Resource estimates leading to reserves that form the basis of pit design should be conservative. Mineral Resources were estimated under the supervision of Mr. Peter Holbek, M.Sc., P.Geo., Copper Mountain's QP responsible for Mineral Resources.

14.2 Resource Estimation Procedures

The resource estimation methodology was similar for all deposits, and involved the following procedures:

- Understanding, to the extent possible, geological controls of mineralization and grade distribution, and determination of domains
- Deposit description and mineralization domains based on geology, structure, and weathering profiles
- Determine suitable block model sizes and extents for each deposit
- Describe drill hole database, validate drill data, and extract the relevant data required for resource estimation
- Analyze the data through univariate and bivariate statistical data analysis Determine what, if any, data conditioning (capping and compositing) is required
- Variography on deposits and deposit domains (required for kriging interpolations)
- Grade interpolation
- Resource, classification, and validation
- Mineral Resource Statement.

14.3 Geological and Mineralization Models and Domains

The following sections describe the criteria for the definition of the geological and mineralization models at the deposits. Domains for grade estimation are based on structural orientation and/or lithological controls on mineralization as well as metallurgical/mineralogical zones related to weathering profiles. The weathering profile for all the copper-gold deposits is reasonably consistent, with an upper zone of oxidized rock generally between 15 m and 25 m in depth, with a relatively sharp boundary between fresh rock or supergene zones, depending upon the deposit. The oxide and supergene zones are defined by observation during core or chip logging and verified by sulphur analyses on a subset of the drill holes within the deposit. Deposit geology and figures describing weathering or supergene domains used for resource estimation are presented in Section 7. Domains defined by structural or lithological orientation are described in Sections 14.5 and 14.8.

14.3.1 Little Eva

Four major structural-lithological domains, separated by faults, have been defined for the Little Eva deposit, each domain with differing orientations of mineralization continuity. Previous workers defined additional subzones of either high- or mid-grade domains based on drill hole copper grades. Recent work, both with the data and limited drill core examination, determined that the subdomain boundaries were gradational and not likely to be visually distinct during mining. Attempts to define the high-grade zones with variography were not successful and therefore these subdomains were not maintained during grade interpolation. A plan view and typical drill sections illustrating the four larger domains (and earlier subdomains) are provided in Figure 7-5 and Figure 7-6.

The upper part of the deposit is oxidized, usually to a depth of 15 m to 25 m, and the transition to sulphide mineralization is quite sharp. The oxidized zone contains copper in native form as well as neotocite (Fe-Mn-Cu mineraloid) and carbonate copper species. Additional testing for recovery of copper from the oxide zone has been carried out, and no economical method of copper extraction has been determined and consequently, the oxide zone is considered to be waste. The oxide zone is present over top of all structural-lithological domains. The contact between the oxide zone and fresh rock was treated as ‘soft’ during interpolation as grade changes across the boundary were minimal.

14.3.2 Turkey Creek

The Turkey Creek deposit was the most recent discovery at the Eva Copper Project and is a copper-only deposit (without gold). Resource estimation of the Turkey Creek deposit was constrained within a stratigraphically controlled grade shell above 0.1% Cu (Figure 7-7 and Figure 7-8). The Turkey Creek deposit occurs as two tabular higher-grade zones separated by a lower grade zone. Although a lower grade internal core has been defined as a domain, these domain boundaries were not used during the interpolation as it may not be possible to segregate this zone during mining. However, interpolated block grades, clearly define the medial low-grade zone indicating that grade interpolation correctly honours drill data, as well as the potential for selective removal during mining, depending upon applied cut-off grade. Changes in orientation of the mineralization on the north end of the deposit resulted in two additional domains.

14.3.3 Blackard and Scanlan

Blackard and Scanlan are very similar deposits geologically, being stratabound with locally deep weathered profiles containing native copper. While these deposits were previously drilled and have historical resource estimates, they were not included in the previous feasibility study due to lower metallurgical recoveries. Additional drilling on Blackard and metallurgical testing on both Blackard and Scanlan, which has defined both, a milling process and a reliable estimates of copper recovery, now allows reserves to be estimated and the incorporation of these deposits into the overall mine plan.

Blackard and Scanlan are nearly identical geologically and metallurgically, both occurring near-surface and within deformed and metamorphosed carbonate rich sediments. Folded stratigraphy and changes in copper mineralization due to weathering require modification to the resource estimation procedures employed for the Little Eva deposit. Both the Blackard and Scanlan deposits appear to occur as thin (10 m) to thick (100 m) bands of mineralization folded into a tight synform and open antiform pair. The deposits contain weathering profiles that include an upper oxide zone, which is treated as waste (although grades are interpolated within the zone), followed by the copper zone where a significant proportion of the copper is contained as fine native copper, followed by a narrow transition zone of mixed metallic copper and sulphide species, and a lowermost sulphide zone. In both deposits, the weathering profile and related native copper zone is much deeper or more extensively developed over the synform part of the deposit areas. The silver content of the sulphide zone is locally significant but was not included in the resource estimates.

The deposits strike northerly and have been subdivided into structural domains based on the dip and/or plunge of the mineralization. An outer shell that reflects interpreted folded stratigraphy, and separates barren rock from mineralization on drill sections, was used to constrain the resource estimates due to linear orientations for interpolation. The deposits were then further subdivided into different domains based on the interpreted strike and dip of the mineralization. Mineralization is folded and curved, whereas interpolation searches are linear, so domain boundaries were generally placed at points of maximum curvature. Where mineralization orientation appeared to change over a short distance a domain boundary was placed where a fault was interpreted. Histograms of assay grades on drill sections display high variability of grade down hole, but in some areas, particularly within the Blackard deposit, alternating high and lower grade bands were noted to align over moderate distances, both on section and along strike these bands were used to guide the orientation of the interpolation search rather than the outline of the grade shell. The boundaries between structural domains and mineralogical domains are treated as soft during resource estimation but are used as hard boundaries for assigning metallurgical recoveries.

14.3.4 Ivy Ann

Ivy Ann is a copper-gold mineralized trend that consists of two deposits hosted within steep, east-dipping zones, with strikes to the north and northeast. The two deposits are separated by 700 m of barren rock and are termed Ivy Ann and Ivy Ann North. The mineralization domain at Ivy Ann includes a main structural zone (Figure 7-13 and Figure 7-15) and two minor hanging wall structures, defined within an outer grade shell at a copper cut-off of 0.1%. At Ivy Ann North there are 14 separate mineralized structures interpreted which were interpolated within a single outer grade shell defined by a copper cut-off of 0.1%.

14.3.5 Lady Clayre

Mineralization at Lady Clayre occurs in a variety of orientations with uncertain geological controls, although it seems that both structure and lithology exert control within a sequence of poly-deformed shales, siltstones, schists, and dolomites. Copper-gold mineralization is coarse-grained and commonly occurs within brecciated rocks. Five zones were defined by Altona based on 0.1% Cu grade shells (Figure 7-9). Confidence in the geological interpretation is limited. Mineralization in the northern part of the deposit strikes northwesterly and dips moderately to steeply to the west, while mineralization to the south strikes northeast and also dips moderately to steeply to the west. The deposit area was divided into two domains based on orientation of mineralization, but neither enclosing grade shells nor smaller subdomains were used due to uncertainty on the placement of mineralization boundaries. A separate domain was used for the oxide zone, which consists of a 15 m to 25 m thick layer with both oxide and carbonate copper species (Figure 7-12).

14.3.6 Bedford

Bedford geology was reinterpreted by Altona in 2016, integrating drill data, surface mapping, high-resolution soil geochemistry, and geophysics into a structural analysis. The confidence in the geological interpretation is moderate to high, based on well-defined local and regional controls on the mineralization geometry. Mineralization outcrops at the surface and has been tested to a depth of 140 m, below which it remains open. The Bedford mineralization is hosted within a steep westerly-dipping shear zone which is 50 m to 120 m wide, striking north-northeast (Figure 7-17 and Figure 7-18). Within the broad shear zone there is an array of mineralized structures with typical widths of 5 m to 12 m, which anastomose but follow the broad overall shear zone trend. Drilling has defined two separate areas of mineralization within the shear zone (Bedford South and Bedford North) where sufficient mineralization is present to be extracted by open pit mining.

Previously, wireframes based on 0.3% Cu grade shells were used to control interpolation; however, because the mineralization is narrow and anastomosing, segregation of ore and waste at lower cut-off grades is difficult. Consequently, the deposit was estimated without constraints using a relatively narrow search ellipse. Resource blocks generated with this methodology should be minable with equipment currently being considered, and potential for higher grades may be possible with careful grade control and mining practices. Both the North and South deposits were subdivided at the base of oxidation, which is an irregular 20 m to 30 m thick layer.

14.3.7 Block Models

Mineral resources are estimated by interpolating composited drill hole grades into a theoretical block model, which divides the space containing the mineralization into rectangles or cubes (blocks). The computer keeps tracks of the blocks by assigning x, y, and z coordinates to the block centroid and knowing the block dimensions. Each block will also be assigned a geological or rock type code by intersecting the block model with 3D solids models of the geology (Figure 14-1). The software used for grade interpolation is GEOVIA Gemcom. The appropriate block size is determined by considering the smallest selective mining unit (SMU), which is a function of either the size and type of mining equipment to be used or the spacing planned for grade control drilling, and the spacing of the data used to interpolate grades into the blocks. Due to differences in the size and shapes of the deposits within the Project area, having different mining equipment available for customized extraction of individual deposits could be beneficial; however, there is no certainty that will happen, and therefore a single block size that could be used for all deposits was investigated. After examining the geology and grade distribution of all the deposits, a 5 m block size was selected, as this size allows for reasonable selectivity, and is convenient for 10 to 15 m benches, with the possibility of 5 m flitches (half bench) when grade distribution and suitably sized equipment indicate that as a preferred option.

Deposit block models are usually laid out as 3D rectilinear shapes that will fully envelop all known mineralization. Details of the deposit block models are provided in Table 14-1.

Table 14-1: 3D Block Model Limits (UTM Coordinates)

Deposit	Direction	Minimum	Maximum	Block Size	No. of Blocks
Little Eva	Easting	410100	411500	5	280
	Northing	7771200	7772900	5	340
	Elevation	-400	180	5	116
Turkey Creek	Easting	412000	413300	5	260
	Northing	7770750	7772750	5	400
	Elevation	-350	250	5	120
Blackard	Easting	411800	413400	5	320
	Northing	7764300	7766800	5	500
	Elevation	-500	400	5	180
Scanlan	Easting	411900	412690	5	158
	Northing	7753650	7755550	5	380
	Elevation	-240	260	5	100
Bedford	Easting	414721	415221	5	100
	Northing	7765598	7768493	5	579
	Elevation	0	210	5	42
Lady Clayre	Easting	409132	410492	5	272
	Northing	7751523	7753283	5	352
	Elevation	-600	400	5	200
Ivy Ann	Easting	425100	427000	5	380
	Northing	7741000	7744600	5	720
	Elevation	-100	280	5	76

All block models are in metric units without any rotation and generally are rectangular shaped, with the long axis to the north and the shorter axis to the east due to the north-south trending deposits. 5 m cubic blocks were used for all deposits, which allows for bench heights of 5 m, 10 m, or 15 m depending on deposit size. Where blocks are cut by a domain boundary (e.g., ore-waste boundary), Gemcom tracks the percentage of the block volume within each domain, which may be required on the narrower zones of mineralization. All block models are in GDA94 / MGA Zone 54 projection.

A variety of information is stored in the block model, including interpolated grades for copper and gold (where present), geological codes, specific gravities (SGs), net smelter return (NSR) calculations, various kriging parameters, metallurgical zones, and block classifications. Block models for Little Eva, Turkey Creek, Blackard, Scanlan, and Bedford are coded according to domains defined by computer solids models built on geological wireframes that represent mineralization boundaries and/or any distinct structural areas or breccia zones. Outer domain boundaries for the Little Eva, Turkey Creek, Blackard, Scanlan, and Bedford deposits were treated as hard, as the boundary is in most cases geological, and any drill data outside the boundary was not used for interpolation of block grades. However, boundaries between domains are soft, and data on either side of the boundary can be used by the interpolation. Blocks are segregated by all domain boundaries. Percent-models are used where blocks straddle domain boundaries to derive the correct volumes and grades. For the Blackard and Scanlan deposits, all drill data were available for interpolation, although not all data were used, as many isolated holes were too distant to have an adjacent hole within the search area, which was a requirement of interpolation protocol. However, only blocks that were inside the geological (or grade) shell were used to sum the resources. The Lady Clayre and Ivy Ann deposits did not have volume models or geology or grade shells, as the drill data were spaced such that the interpolations were adequately constrained by search area.

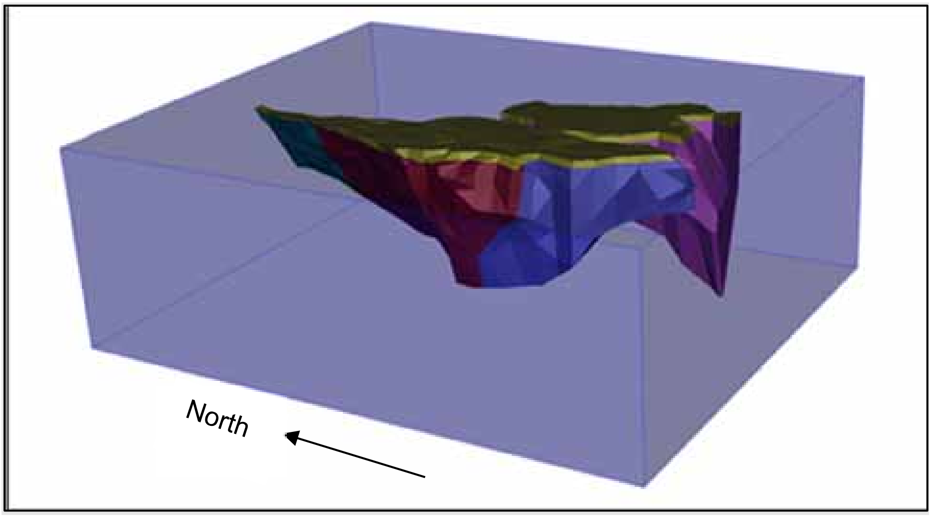
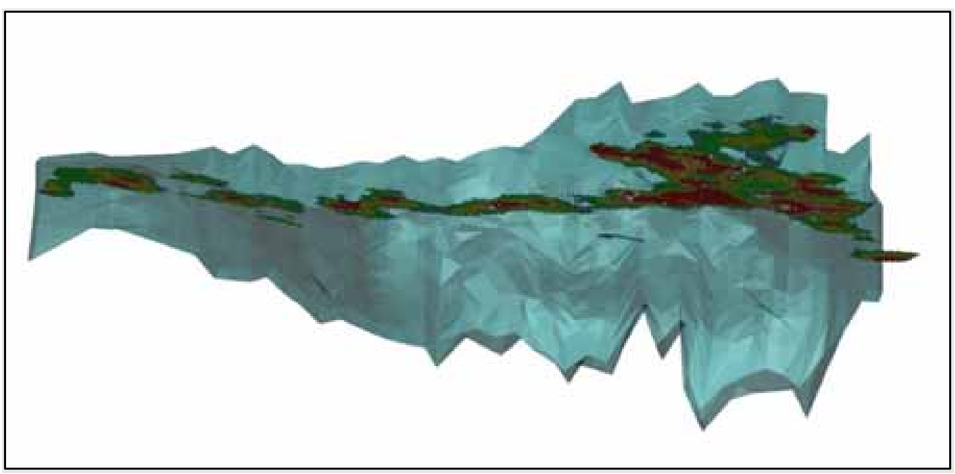


Figure 14-1: Little Eva Block Model Outline in Blue Containing the Structural Estimation Domains in Various Colours and the Oxide Layer in Yellow



Note: Built for the Blackard deposit showing block grades by colour for the 110 elevation.
Image view is Eastwards with North to the left

Figure 14-2: Isometric View of the Constraining Volume (solids model)

14.4 Database and Statistical Analysis

14.4.1 Drill Hole Database

The Copper Mountain Mining Corp. (CMMC)-Altona drill hole database is stored on the company server and is in Access format. Data for each of the deposits was uploaded to a Gemcom workspace, where it was reviewed and analyzed. Little Eva, Turkey Creek, Blackard, and Scanlan databases were uploaded directly from the previously defined file structure used for the 2012–2015 Optiro resource estimates and any data from recent drilling was added to the appropriate data base. Data quality was reviewed and combined with Altona's extensive previous validation work by third-party consultants, the data was determined to be of high quality, and valid for use in resource estimation. Standard checks (missing intervals, missing holes, overlapping intervals) did not reveal any errors in the database, although there were a small number of copper assays without a corresponding gold assay (Table 14-2). The Project database includes collar, survey, assay, and lithological information, as well as drill hole type, year drilled, and company information from the various historical drill campaigns. Where data was loaded directly from the Project Access database, CMMC used the same dataset that the previous Optiro and Altona resource estimates had used, (see Table 14-2). Both, diamond and RC drilling have taken place throughout all the deposits by many companies, including Universal, Dominion, Bruce Resources, PanAust, Xstrata, and Altona. A small amount of drilling for due diligence and/or to collect metallurgical sample material was carried out by Sichuan Railway Investment Group (SRIG) and CMMC in between 2016 and 2018 and an 18-hole RC program on the Blackard deposit was completed in 2019. Table 14-2 gives the breakdown of the drill data by company, the number of drill holes, and the years that the drilling occurred in all the current resource areas.

The drilling history of the Eva Copper Project dates to the late 1970s, when CRAE began drilling in the Little Eva and Lady Clayre areas (Figure 14-3, Table 14-2). Since then, numerous campaigns of RAB, RC, and diamond drilling have taken place throughout the Project area and on all the deposits by many companies including: Universal, Dominion, Bruce Resources, Pan Australian, Xstrata, and Altona. A small amount of drilling for due diligence grade confirmation and to obtain fresh samples for metallurgical testing was carried out by SRIG (2016-2017) and Copper Mountain in 2018 and 2019.

Most of the drilling on the Project was focused on the Little Eva (36%) (Figure 14-4) and Blackard (28%) deposits while Lady Clayre has 11% of total drilled metres, followed by Scanlan, Bedford, Ivy Ann, and Turkey Creek, with 9%, 6%, 6%, and 4%, respectively. Much of the RAB drilling was for exploration outside of the deposit areas, and due to possible contamination issues with RAB samples, no RAB holes were used in the resource estimations.

Table 14-2: Summary of Exploration Drilling by Company

Deposit	Year	Company	Hole Type	Hole Count	Metres	% of Total
Little Eva	1978–1996	CRAE	DD	6	2,330	35%
			RC	61	5,293	
	2002–2006	Universal	RC	281	37,855	
			DD	30	4,037	
	2006	Xstrata	DD	2	984	
	2011–2018	Altona	RC	102	20,899	
			DD	11	2,572	
	2018	CMMC	DD	4	202	
Turkey Creek	1993	CRAE	RC	2	218	4%
	2011	Xstrata	RC	2	300	
	2012–2015	Altona	RC	49	7,296	
			DD	5	404	
	2019	CMMC	DD	1	132	
Blackard	1991–1995	CRAE	DD	19	4,769.6	26%
			RC	8	1,120	
			PERC	6	613	
	2002	Bolnisi Logistics	DD	7	927.8	
			RC	121	13,558	
	2005–2009	Universal	DD	46	12,419	
			RC	117	13,746	
	2011	Altona	DD	3	548	
			RC	21	4,049	
	2019	CMMC	RC	18	2,695	
Scanlan	1991–1995	CRAE	RC	97	7,553	9%
			DD	5	1,635.9	
			AC	3	110	
	2002	Bolnisi Logistics	RC	2	397	
	2005–2006	Universal	RC	45	5,358	
			DD	11	1,803.2	
	2007–2008	Xstrata	DD	2	798.2	
	2010	Universal	RC	7	1,324	
Bedford	1990	CRAE	RC	5	420	6%
	2003–2009	Universal	RAB*	43	1,680	
			RC	97	9,762	
			DD	1	160	
	2015	Altona	DD	1	36	
Ivy Ann	1992–1993	Dominion	RAB*	26	863	7%
			RC	15	1,591	
	1995	Bruce Resources	RC	11	1,084	
	1995–1996	Pan Australian	RC	10	1,268	
			RAB	44	1,972	

Deposit	Year	Company	Hole Type	Hole Count	Metres	% of Total
	2003–2009	Universal	RC	18	2,205	
	2011–2012	Altona	RC	27	5,448	
Lady Clayre	1978–1998	CRAE	RAB*	50	471	13%
			RC	46	5,477	
			DD	30	7,994	
	2002–2009	Universal	RAB	39	1,913	
			RC	40	4,967	
			DD	2	153.9	
	2011–2012	Altona	RC	27	5,188	
Total				**1,626**	**208,600**	

Notes: DD = diamond drilling, RC = reverse circulation, RAB = rotary air blast
* denotes holes not used in resource estimates.

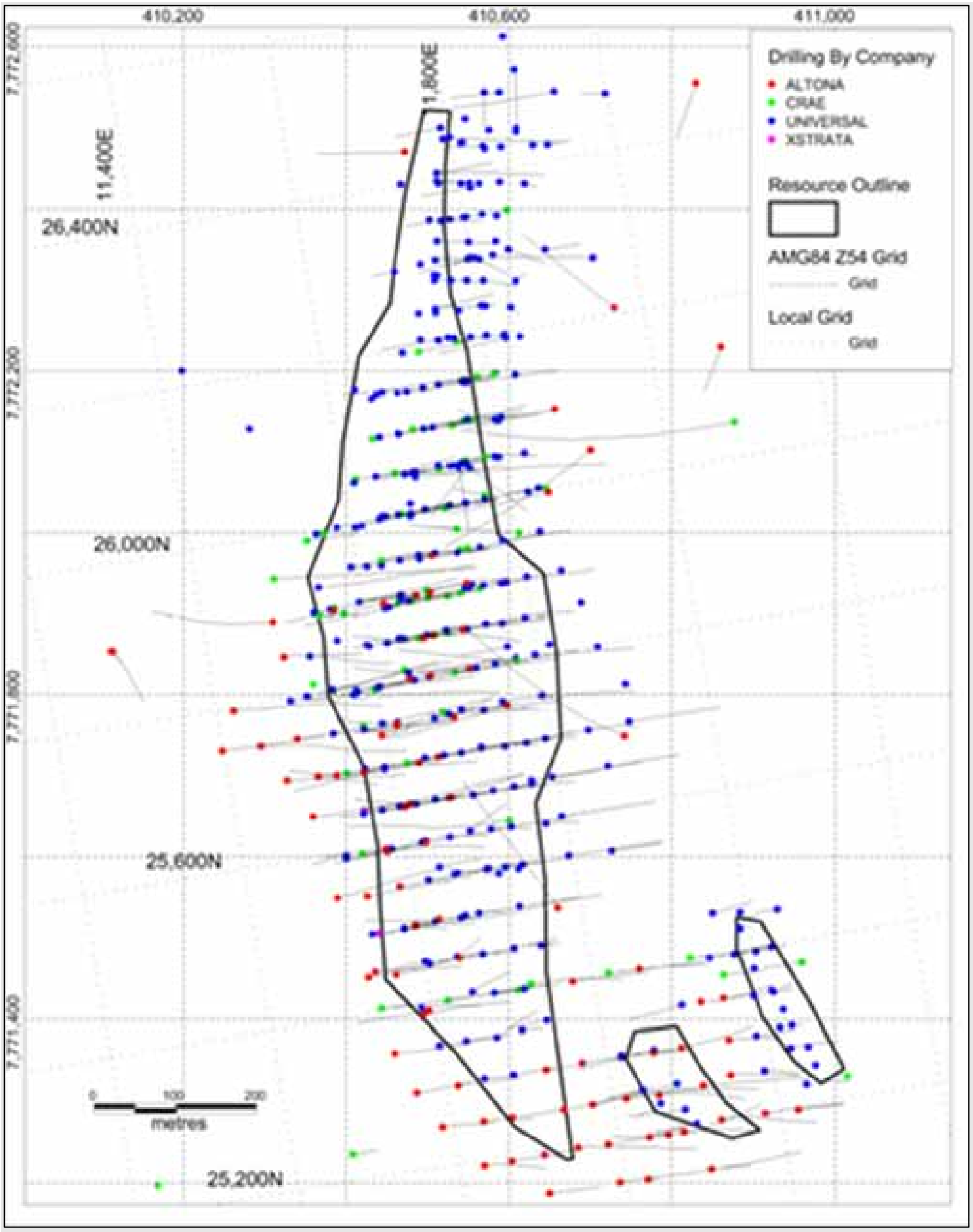
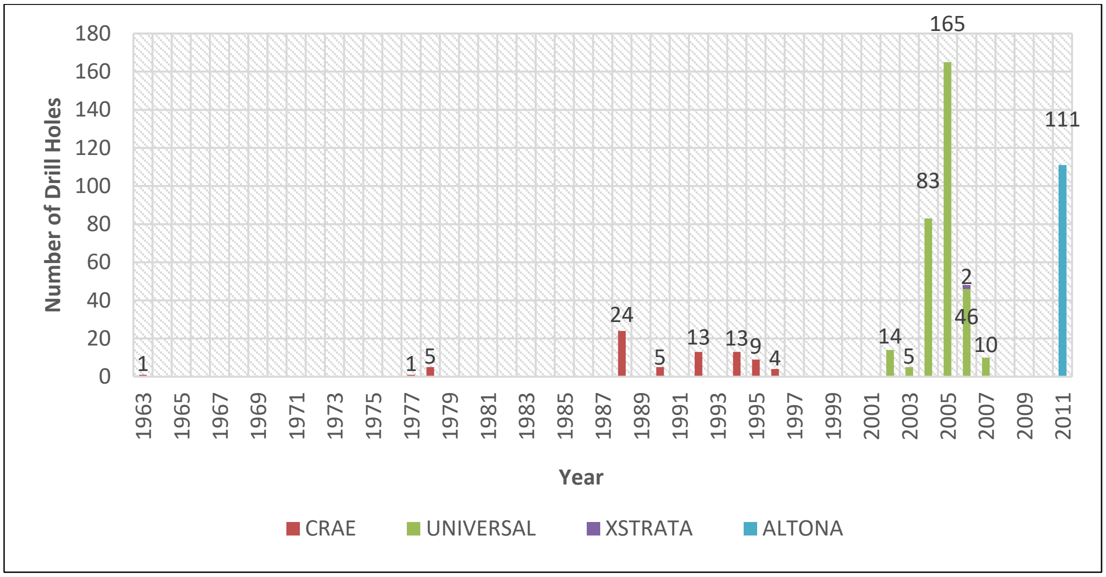


Figure 14-3: Little Eva Drill Collar Plan by Company



Note: Chart displaying up to year 2011, which is most of the Little Eva drilling. The chart does not show limited drilling, predominately for metallurgical samples in 2015 and 2018

Figure 14-4: Number of Little Eva Drill Holes by Year and Company

14.4.2 Deposit Assay Data Statistics

Assay datasets for each deposit were examined by simple univariate statistics to provide a basic understanding of ranges, distribution, and variance to determine the appropriate methods of resource estimation. In some cases, outlying holes which did not intersect the area of mineralization were removed prior to statistical analysis. A summary of assay statistics for each deposit is provided in Table 14-3, and selected histograms of assay data and composites are presented in the various figures and tables that follow in this section.

The Little Eva deposit has lognormal distributions of both copper and gold, with a high but acceptable coefficient of variation (CoV), which becomes even more subdued following compositing at 2.5 m. Turkey Creek mineralization is a much smaller dataset, and is unusual in that it is negatively skewed, with the number of samples increasing towards higher grades, which is likely a function of visually distinct mineralized zones favouring sampling within the mineralization. The Bedford, lady Clayre, and Ivy Ann deposits all have high maximums and correspondingly high CoVs, with low median values due to multiple relatively narrow zones of mineralization separated by non-mineralized material. The Blackard and Scanlan deposits both have log-normal distributions with relatively low CoV's and have similar statistics to each other with slightly higher median and mean grades in the Scanlan deposit.

Table 14-3: Summary of Assay Statistics by Deposit

Deposits	Statistics	Raw Assays*|Uncapped		Assays in Estimate	
		Cu (%)	Au (g/t)**	Uncapped Cu (%)	Capped Au (g/t)
Little Eva	Count	46,678	46,321	45,651	45,409
	Mean	0.36	0.06	0.36	0.06
	Median	0.18	0.03	0.18	0.03
	Minimum	0.00	0.00	0.00	0.00
	Maximum	18.89	4.4	18.89	4.4

Deposits	Statistics	Raw Assays*|Uncapped		Assays in Estimate	
		Cu (%)	Au (g/t)**	Uncapped Cu (%)	Capped Au (g/t)
	Std. Dev.	0.63	0.12	0.63	0.12
	CoV	1.76	1.86	1.75	1.84
Turkey Creek	Count	7,207	-	2,591	-
	Mean	0.19	-	0.46	-
	Median	0.03	-	0.34	-
	Minimum	0	-	0	-
	Maximum	4.5	-	4.5	-
	Std. Dev.	0.33	-	0.41	-
	CoV	1.71	-	0.88	-
Bedford	Count	9,446	4,167	4,942	2,569
	Mean	0.37	0.11	0.25	0.12
	Median	0.07	0.03	0.04	0.04
	Minimum	0.00	0.01	0.00	0.01
	Maximum	11.5	6.46	11.5	1.5
	Std. Dev.	0.84	0.28	0.68	0.21
	CoV	2.29	2.6	2.69	1.8
Lady Clayre	Count	19,157	19,218	19,157	19,218
	Mean	0.16	0.07	0.16	0.07
	Median	0.03	0.01	0.03	0.01
	Minimum	0.00	0.00	0.00	0.00
	Maximum	20.7	45.1	20.7	45.1
	Std. Dev.	0.48	0.49	0.48	0.49
	CoV	3.09	7.43	3.09	7.43
Ivy Ann	Count	11,458	11,458	11,458	11,458
	Mean	0.17	0.03	0.17	0.03
	Median	0.03	0	0.03	0
	Minimum	0.00	0.00	0.00	0.00
	Maximum	23.5	3.18	23.5	3.18
	Std. Dev.	0.51	0.08	0.51	0.08
	CoV	3.09	3.22	3.09	3.22
Blackard	Count	33,706	-	33,706	-
	Mean	0.269	-	0.269	-
	Median	0.11	-	0.11	-
	Minimum	0.02	-	0.02	-
	Maximum	6.66	-	6.66	-
	Std. Dev.	0.395	-	0.395	-
	CoV	1.469	-	1.469	-
Scanlan	Count	9,424	-	9,424	-
	Mean	0.31	-	0.31	-
	Median	0.138	-	0.138	-
	Minimum	0	-	0	-
	Maximum	6.85	-	6.85	-
	Std. Dev.	0.46	-	0.46	-
	CoV	1.51	-	1.51	-

Note: *Assay data from both sulphide and oxide zones. **Au assays for Little Eva capped at 4.4 g/t in this dataset.

More detailed analysis was carried out on the Little Eva, Blackard, and Scanlan deposits to evaluate potential for data bias between drill hole type, company, and drill hole orientation. Comparisons of statistics between exploration programs by company are shown in Table 14-4. In general, the statistics are quite similar, except for the mean grade, which is lower for the Altona holes due to more drilling around the edges of the deposit, as can be observed in Figure 14-5 which is a drill hole plan with collars colour-coded by company. Similarly, Table 14-5 compares basic statistics between reverse circulation and diamond drill holes (DDH); the higher mean and medians for the DDHs are a result of the DDHs being preferentially drilled in the north-central, higher grade area of the deposit as illustrated in Figure 14-5.

14.4.2.1 Little Eva Deposit

Drill hole data for the Little Eva deposit was examined for any form of bias related to different drill programs by different companies, or differing drill equipment or drill hole orientations. While differences are noted in Table 14-4 and Table 14-5, it is believed these differences are more reflective of the location of the drill holes as opposed to any inherent bias. The database is deemed good for resource estimation.

Table 14-4: Summary of Cu Assay Statistics for Little Eva by Company Drill Data

	CRAE	Universal	Altona
Count	1,760	30,136	14,185
Mean	0.41	0.385	0.283
Median	0.19	0.19	0.164
Minimum	0	0	0
Maximum	16.8	18.9	13.9
Std. Dev.	0.79	0.68	0.45
Log Variance	1.61	1.67	1.94
CoV	1.96	1.75	1.59

Table 14-5: Summary of Basic Statistics for RC vs. Diamond Drill Hole Assays for Little Eva

	RC	DD
Count	42,596	3,485
Mean	0.34	0.48
Median	0.18	0.22
Minimum	0	0
Maximum	18.9	13.15
Std. Dev.	0.6	0.83
Log Variance	1.77	1.71
CoV	1.75	1.73

A review of drill hole orientation was undertaken on the Little Eva deposit to assess for any grade bias in the data. Drill holes were partitioned based on either easterly orientations (azimuth 50–140), westerly orientations (azimuth 230–320) or vertical holes (dip ≥ -80). A very small sub-set of holes (three) were drilled towards the south and are not included. Statistics for copper assays based on drill hole orientation is provided in Table 14-6, and indicates that there is little difference between westerly-inclined drill holes and easterly-inclined drill holes. Vertical drill holes have a higher mean grade, which is likely more related to a concentration of these holes in the higher-grade central and northerly parts of the deposit. Examination of the detailed drill sections indicates that mineralization is not systematically vertically oriented, and therefore vertical drill holes are unlikely to produce grade bias.

Table 14-6: Basic Statistics for Capped Assays* by Domain at Little Eva

	North		Central		South		South-East		Total	
	Cu%	Au g/t	Cu%	Au g/t	Cu%	Au g/t	Cu%	Au g/t	Cu%	Au g/t
Count	1,466		19,214		13,947		4,467		39,094	
Mean	0.802	0.093	0.445	0.067	0.265	0.062	0.291	0.052	0.377	0.064
Median	0.29	0.05	0.228	0.03	0.163	0.03	0.173	0.03	0.191	0.03
Mode	0.001	0.001	0.001	0.005	0.06	0.005	0.001	0.005	0.001	0.005
Minimum	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001
Maximum	9.71	4	18.89	1.5	16.09	1.5	11.73	1	18.89	4
Std. Dev.	1.22	0.19	0.76	0.12	0.39	0.1	0.44	0.08	0.656	0.115
CoV	1.53	2.05	1.71	1.85	1.48	1.63	1.51	1.54	1.74	1.79

Note: *Assays were normalized to 1 m to avoid length weighted bias in the dataset. Gold values in the data were capped at 1.5 g/t except for five samples in the north zone which were capped at 4 g/t by Optiro-Altona in May 2014.

Table 14-7: Cu% Assay Statistics Based on Drill Hole Orientation at Little Eva

	East Dip	West Dip	Vertical
Count	30,707	6,618	8,537
Mean	0.32	0.37	0.45
Median	0.17	0.18	0.21
Minimum	0.00	0.00	0.00
Maximum	16.80	16.09	18.89
Std. Dev.	0.54	0.69	0.78
CoV	1.70	1.87	1.72

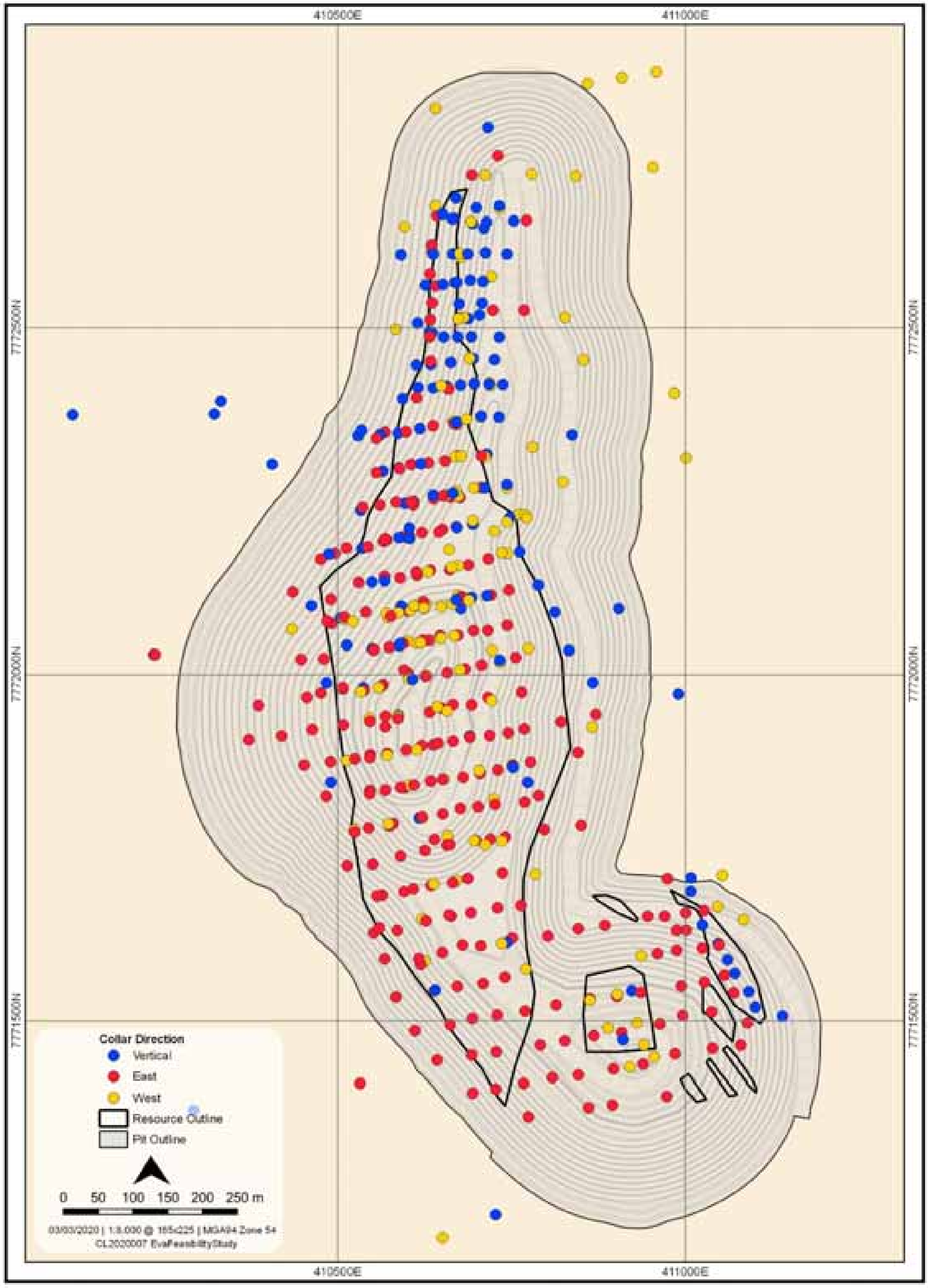
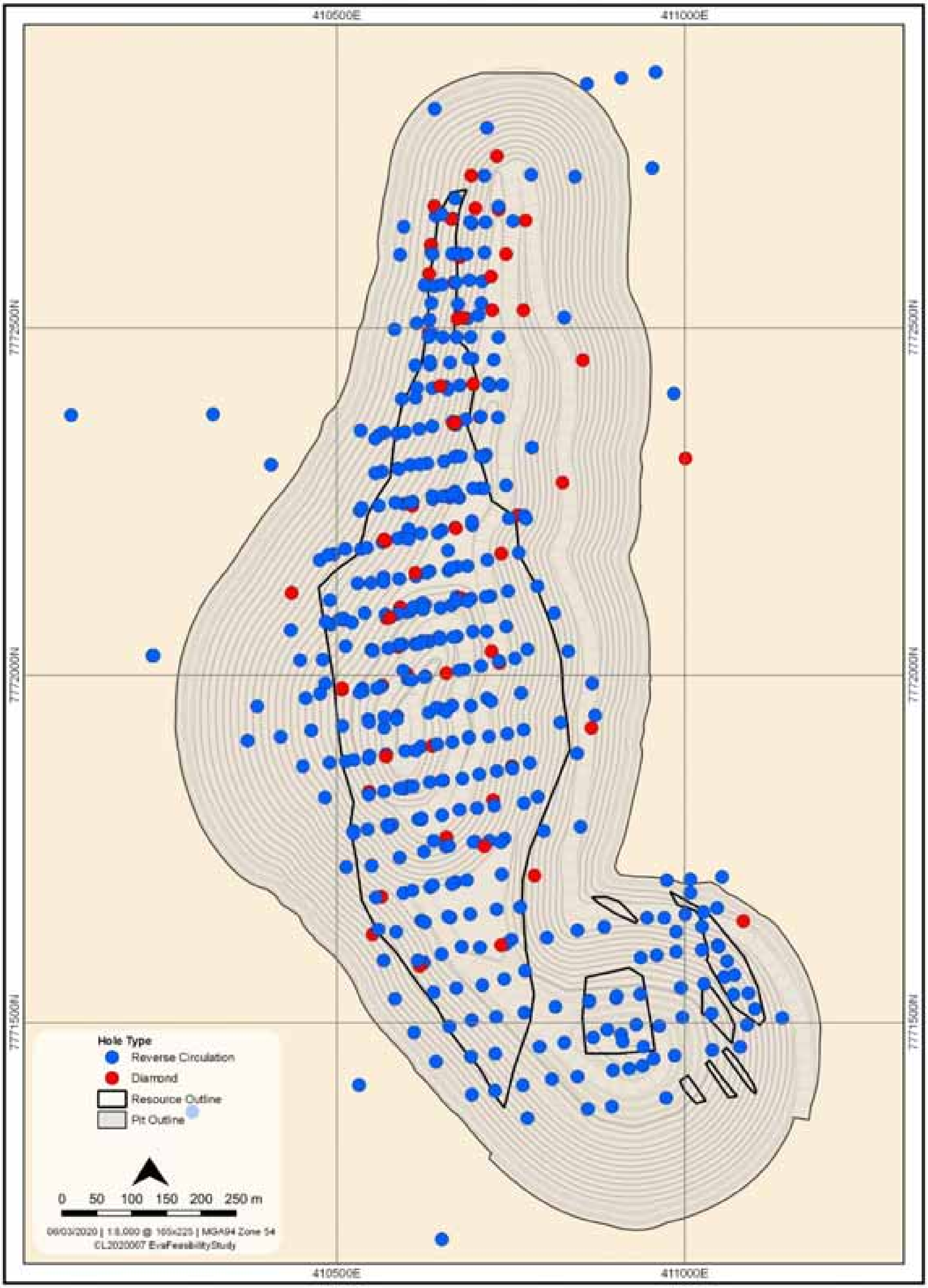


Figure 14-5: Little Eva Drill Collar Plan with Drill Holes Colour Coded by Orientation



Note: The concentration of vertical holes in the northern, higher-grade end of the deposit.

Figure 14-6: Plan View of Drill Collars Colour-Coded by Drill Type

14.4.2.2 Blackard Deposit

A combination of drilling was used on the Blackard deposit and potential for bias between drill types was examined. Additionally, basic statistics between domains were examined to support the use of soft domain boundaries during interpolation. Examples of assay data statistics for Blackard domains are provided below, whereas examples of statistical analysis of composited data for the various deposits is provided in Section 14.5.3.

Basic statistical analysis of copper assays was initially carried out on each mineralogical/structural domain; however, a better comparison is provided by data above a very low-grade cut-off value as shown in Table 14-8. For statistical analysis two structural or orientation domains were defined for the Blackard deposit: variable west dipping (tilt) and horizontal (flat) mineralization, which was a simplification of a curvilinear deposit (a vertical domain was defined later for resource estimation). Domains that are evenly divisible by 10 are dipping domains on the western and northern parts of the deposit and those that are evenly divisible by 5 are flat-lying domains on the eastern side. The oxide zone typically has lower grades than the other zones which is as expected since copper has been leached from this zone; however, the leaching is more pronounced above the dipping mineralization than in the flat-lying mineralization. A similar effect, but less pronounced is also evident in the copper zone, whereas there is no statistical difference between the dipping and flat-lying sulphide mineralization. The relatively small differences in statistical measures between the zones supports the use of soft boundaries.

Table 14-8: Cu% Assay Statistics by Resource Domain for the Blackard Deposit

Zone	Oxide		Native Cu		Transition	Sulphide	
Domains	**10 (Tilt)**	**15 (Flat)**	**20 (Tilt)**	**25 (Flat)**	**30 (Tilt)**	**40 (Tilt)**	**45 (Flat)**
Blackard Deposit Assay Statistics (All assays)							
Count	4,118	1,630	10,879	1,786	1,167	11,982	2,144
Mean	0.151	0.344	0.381	0.504	0.336	0.175	0.162
Median	0.070	0.260	0.230	0.410	0.151	0.035	0.028
Mode	0.030	0.030	0.030	0.040	0.020	0.001	0.005
Minimum	0.000	0.001	0.000	0.003	0.001	0.000	0.000
Maximum	2.500	1.620	6.660	3.790	5.110	5.530	6.660
Std. Dev.	0.209	0.297	0.429	0.422	0.542	0.350	0.416
CoV	1.385	0.863	1.125	0.837	1.614	2.002	2.566
Assays at, or >0.05% Cu Cut-off							
Count	2,610	1,448	9,243	1,664	866	5,366	843
Mean	0.224	0.384	0.445	0.540	0.445	0.375	0.392
Median	0.131	0.300	0.310	0.450	0.250	0.206	0.205
Mode	0.050	0.150	0.050	0.150	0.040	0.050	0.050
Minimum	0.050	0.050	0.050	0.050	0.050	0.050	0.050
Maximum	2.500	1.620	6.660	3.790	5.110	5.530	6.660
Std. Dev.	0.233	0.291	0.436	0.416	0.591	0.449	0.593
CoV	1.042	0.758	0.979	0.771	1.329	1.197	1.514

Basic statistics were also examined by drill type for Blackard. The differing statistics between core drilling and RC drilling is believed to reflect the different locations of the drill holes (Table 14-9). Diamond drill holes were used for deep step-outs, and therefore drilled much greater distances in weakly mineralized or unmineralized areas than the RC drilling, which is more concentrated within the deposit area. There are no significant statistical differences when grades within the deposit shell were compared for the two drill types.

Table 14-9: Cu% Assay Statistics by Drill Type for the Blackard Deposit

Drill Hole Type	Core	RC
Count	13,069	21,096
Mean	0.220	0.291
Median	0.046	0.140
Minimum	0.000	0.000
Maximum	6.660	6.660
Std. Dev.	0.392	0.390
CoV	1.778	1.341

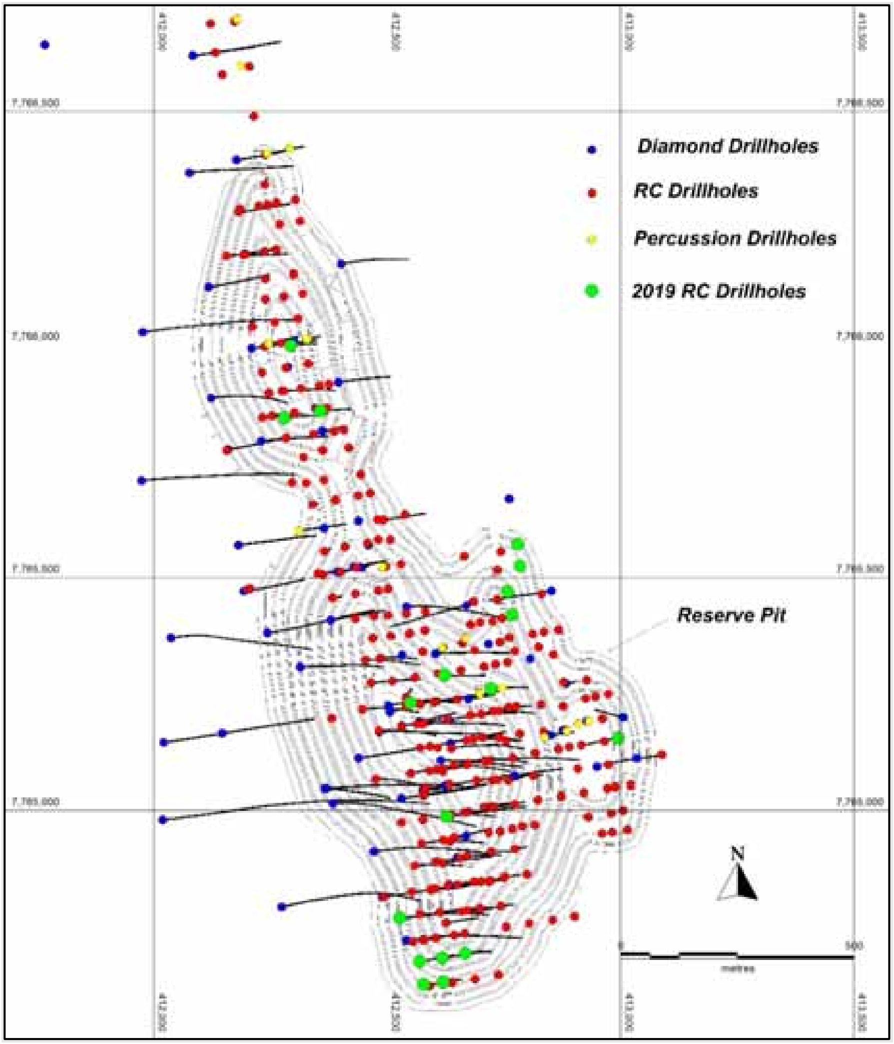


Figure 14-7: Plan View of Drill Collars, Colour-Coded by Type for the Blackard Deposit with Reserve Pit Shell Shown

14.4.2.3 Scanlan Deposit

Most of the drilling on the Scanlan deposit was by RC, with only the deeper (down-dip) and metallurgical holes completed by diamond drilling, as displayed by the drill plan in Figure 14-8. Therefore, a significant amount of the diamond drilling was peripheral to the ore zone. Grades within metallurgical holes drilled proximal to RC holes are similar and no bias between drill types has been detected.

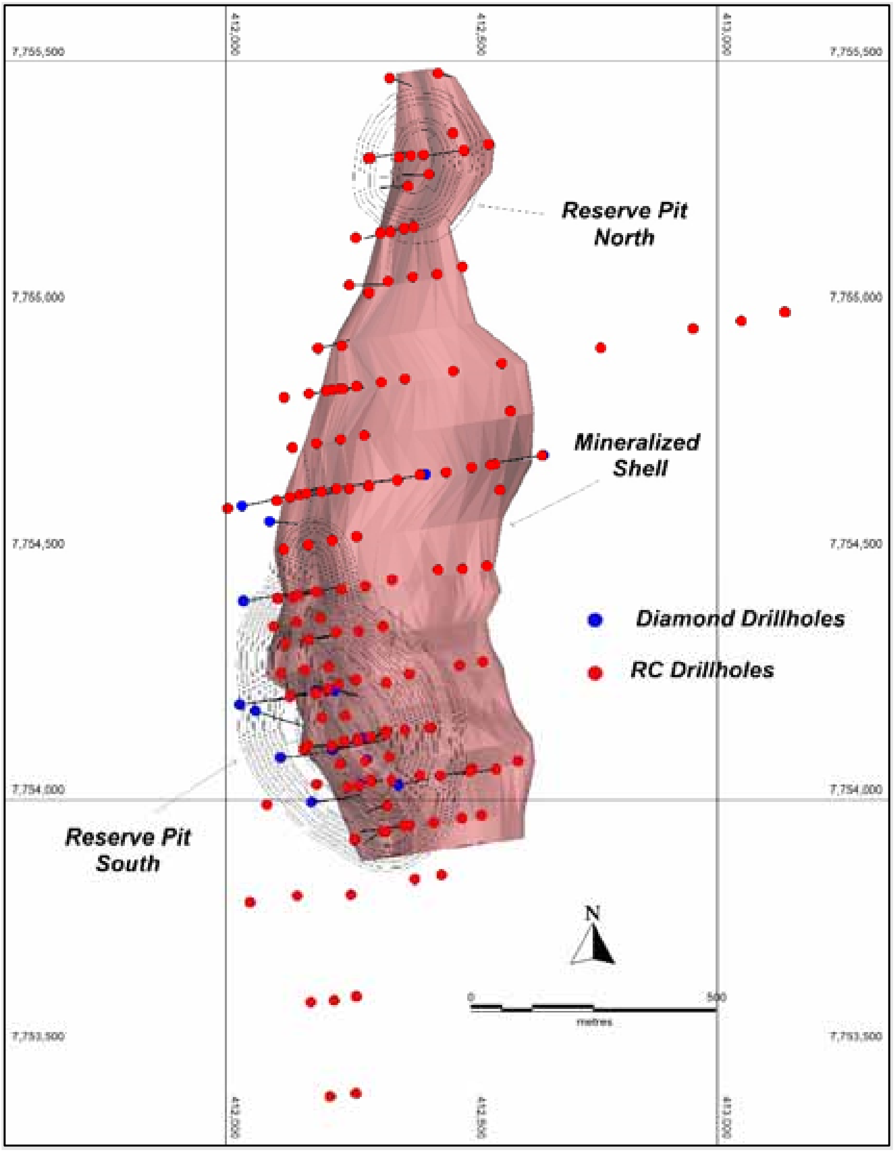


Figure 14-8: Plan View of Drill Collars for the Scanlan Deposit Showing the Resource Shell and Reserve Pit Outlines

14.4.3 Data Conditioning and Assay Composites

Data populations were examined using histograms and probability plots. Histograms indicate that distribution of both copper and gold grades are log-normal, with varying amounts and directions of skewness between deposits. Compositing of all drill hole assays to equal lengths is required for interpolation. The choice of composite length is determined by raw data distribution and block size: composite lengths must be less than the block size, and half the block height is a common and convenient selection. Compositing the predominately 1 m assays to a half-block length of 2.5 m smooths out the histograms, and indicates a reasonably uniform distribution of values, apart from spikes at, or near, analytical detection limits as illustrated in Figure 14-9 and Figure 14-10. Cumulative probability plots were examined to determine whether capping would be required. (Note that data analysis was done on a dataset where gold had already been capped to a maximum value of 1.5 g/t, except for 5 values that were capped at 4.0 g/t with resulting values which ranged from 1.8 g/t 4.0 g/t). Basic statistics, for 2.5 m composites by domain for the seven deposits are provided in Table 14-10 through Table 14-16, and can be compared with the raw data statistics Table 14-3. Some high assay values (> 6.5% Cu) in the drill data from Little Eva and Lady Clayre remained after compositing, however, these values were neither random nor isolated anomalies but part of a continuous population distribution. The number of high-value composites is very small (for example in Little Eva deposit the number of composites >4.0%Cu is less than 0.0026% of the total number of composites) and would have a negligible impact on resource values. Consequently, capping of copper assays or composites was not required. The single very high assay in the Lady Clayre deposit was from a very narrow sample and was reduced to 4% Cu by the compositing process.

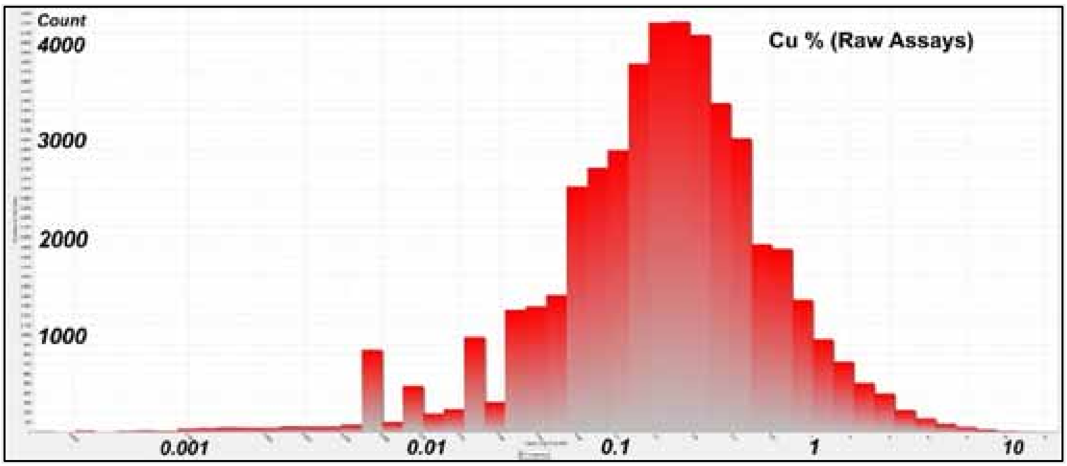
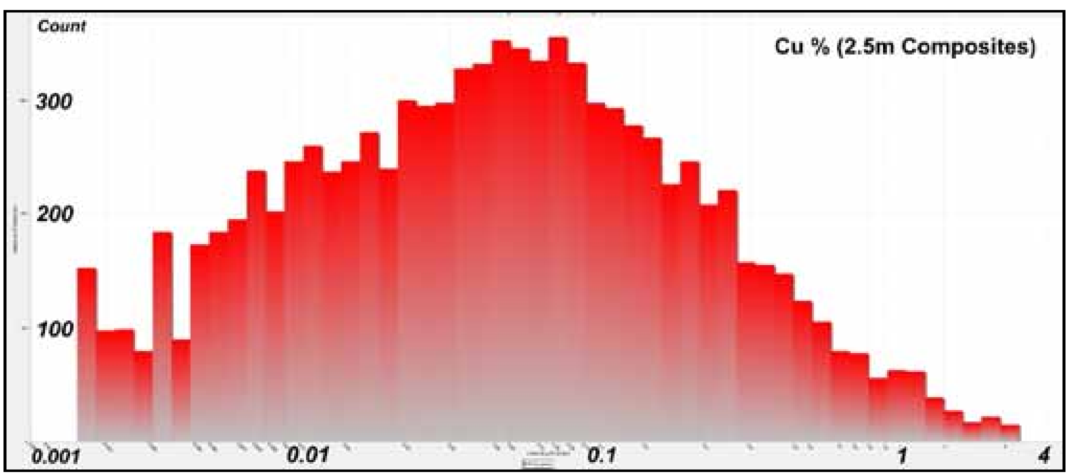


Figure 14-9: Log Histogram for Raw Assay Data from Little Eva Deposit



Note: Bold values below the histograms are arithmetic.

Figure 14-10: Log Histogram of 2.5 m Copper Composites, Little Eva Deposit

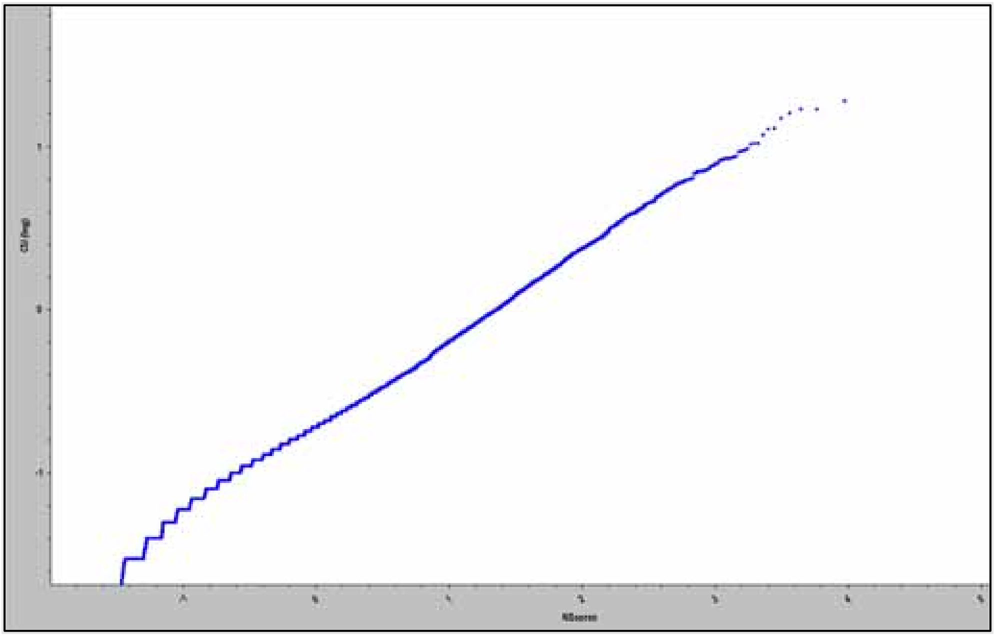


Figure 14-11: Cumulative Probability Plot for Cu Assays, Little Eva

Table 14-10: Basic Statistics for 2.5 m Composites by Domain at Little Eva

	North		Central		South		South-East		Total	
	Cu %	Au g/t	Cu %	Au g/t	Cu %	Au g/t	Cu %	Au g/t	Cu %	Au g/t
Count	566		7,827		5,581		2,492		16,466	
Mean	0.93	0.11	0.45	0.07	0.27	0.06	0.23	0.05	0.371	0.064
Median	0.598	0.074	0.263	0.037	0.186	0.04	0.142	0.028	0.216	0.037
Mode	0.001	0.001	0.001	0.005	0.14	0.005	0.005	0.005	0.001	0.005
Minimum	0.001	0.001	0.001	0.001	0.001	0.000	0.000	0.001	0.000	0.000
Maximum	6.892	2.83	11.476	1.788	8.566	1.016	4.816	0.694	11.476	2.834
Std. Dev.	1.00	0.16	0.63	0.10	0.30	0.08	0.31	0.06	0.535	0.091
CoV	1.07	1.52	1.39	1.46	1.13	1.24	1.34	1.27	1.44	1.42

The Turkey Creek deposit has been defined by 51 RC, and 5 DDH for a total of 7,814 m. Twenty-two of the holes, and an extension to one hole (together totalling 2,778 m), were completed during November and December 2014. The mineralization is strongly tabular and stratabound, striking north-south and dipping east at 60°. At the northern end of the deposit, the strike of the mineralization swings sharply towards the east, and dips steeply south (Figure 14-19). Basic statistics of composite assays by domain are provided in Table 14-11. Although the number of composites is low in comparison to the to the other deposits, the grade continuity of mineralization within the narrow, tabular zones is sufficient for defining Measured and Indicated resources.

Table 14-11: Basic Statistics for 2.5 m Composites by Domain at Turkey Creek

	South Zone Sulphide (Cu)	North Zone Sulphide (Cu)	North Fold Zone Sulphide (Cu)	Oxide Zone All (Cu)
Count	630	67	77	282
Mean	0.440	0.478	0.280	0.529
Median	0.369	0.485	0.233	0.419
Mode	0.318	0.418	0.253	0.432
Minimum	0.001	0.002	0.015	0.002
Maximum	1.824	1.12	1.101	2.521
Std. Dev.	0.303	0.284	0.210	0.412
CoV	0.688	0.594	0.751	0.778

Table 14-12: Basic Statistics for 2.5 m Composite Grades in Blackard Deposit by Structural and Mineralogical Domains

Domain	**15**	**20**	**25**	**30**	**35**	**40**	**45**	**50**	**60**	**70**
Structural Zone		**Inclined**			**Flat**			**Vertical (North)**		
	Oxide	**Copper**	**Transition**	**Sulphide**	**Copper**	**Transition**	**Sulphide**	**Copper**	**Transition**	**Sulphide**
Cu% by Domain Sample Above 0.05% Cu Cut-off										
Count	2,889	5,783	1,109	502	77	2,255	440	684	68	645
Mean	0.279	0.440	0.522	0.438	0.545	0.360	0.329	0.354	0.386	0.263
Median	0.178	0.318	0.428	0.233	0.278	0.210	0.188	0.234	0.253	0.134
Mode	0.050	0.050	0.210	0.066	0.404	0.050	0.054	0.050	0.344	0.064
Minimum	0.050	0.050	0.050	0.050	0.050	0.050	0.050	0.050	0.052	0.050
Maximum	1.896	6.216	2.354	3.906	2.908	3.266	4.868	3.954	2.428	2.212
Std. Dev.	0.258	0.409	0.398	0.574	0.636	0.402	0.480	0.354	0.421	0.310
CoV	0.924	0.931	0.761	1.311	1.167	1.119	1.461	1.001	1.091	1.177
Cu% by Domain Inside Deposit Shell										
Count	1,877	4,634	1,068	422	61	1612	293	522	57	351
Mean	0.373	0.521	0.534	0.486	0.651	0.426	0.383	0.425	0.411	0.374
Median	0.296	0.436	0.443	0.299	0.402	0.283	0.232	0.299	0.262	0.260
Mode	0.001	0.001	0.001	0.001	0.404	0.001	0.001	0.296	0.202	0.112
Minimum	0.001	0.001	0.001	0.001	0.020	0.001	0.001	0.033	0.024	0.008
Maximum	1.896	6.216	2.354	3.906	2.908	3.082	4.868	3.954	2.428	1.894
Std. Dev.	0.276	0.418	0.401	0.600	0.674	0.432	0.549	0.373	0.454	0.356
CoV	0.740	0.802	0.752	1.235	1.035	1.015	1.431	0.879	1.105	0.951

Note: *Refer to Figure 7-10 for mineralogical domains and Figure 14-22 for structural domains.

Table 14-13: Basic Statistics for Composites by Mineralogical Zone for the Scanlan Deposit

	Oxide	Copper Zone	Transitional	Sulphide Zone*	Total
Count	1,267	4,098	254	1,244	6,933
Mean	0.152	0.412	0.211	0.108	0.299
Median	0.076	0.238	0.072	0.030	0.130
Mode	0.01	0.001	0	0	0
Minimum	0	0	0	0	0
Maximum	1.58	5.41	2.08	2.66	5.41
Std. Dev.	0.20	0.52	0.37	0.24	0.45
CoV	1.33	1.26	1.77	2.25	1.50

Note: * Includes all drilling below the deposit.

The mineralization at the Bedford deposit is separated into two deposits, Bedford North and South, which are 600 m apart and lie within the same structure. Previously, narrow high-grade structures were modelled using a sectional approach for the Bedford estimates to constrain the resource. However, CMMC only constrained the interpolation within the broader low-grade envelope (modelled by Altona) using a larger block size of 5 m^3. Blocks were interpolated based on two passes, which generated both Indicated and Inferred resources. Due to data spacing and a limited number of composites, a Measured pass was not interpolated.

Table 14-14: Basic Statistics for 2.5 m Composites by Domain at Bedford

	Primary		Oxide	
	Cu	Au	Cu	Au
Count	2,486	2,486	799	799
Mean	0.25	0.07	0.24	0.06
Median	0.06	0.01	0.06	0.01
Minimum	0.00	0.00	0.00	0.00
Maximum	6.07	4.49	7.5	2.4
Std. Dev.	0.54	0.19	0.60	0.17
Coefficient of Variance	2.18	2.82	2.51	2.62

Mineralization at Lady Clayre has been separated into two zones, East and West, which together form a V shape. The West zone mineralization trends north-south and was previously interpreted into 11 different domains defined by modelled grade shells. The East zone was also split into multiple domains. As with Bedford, CMMC choose to model just the two zones within the low-grade envelope based on a 0.1% Cu cut-off and use different search parameters for each zone based on the interpreted generalized trends to the mineralization.

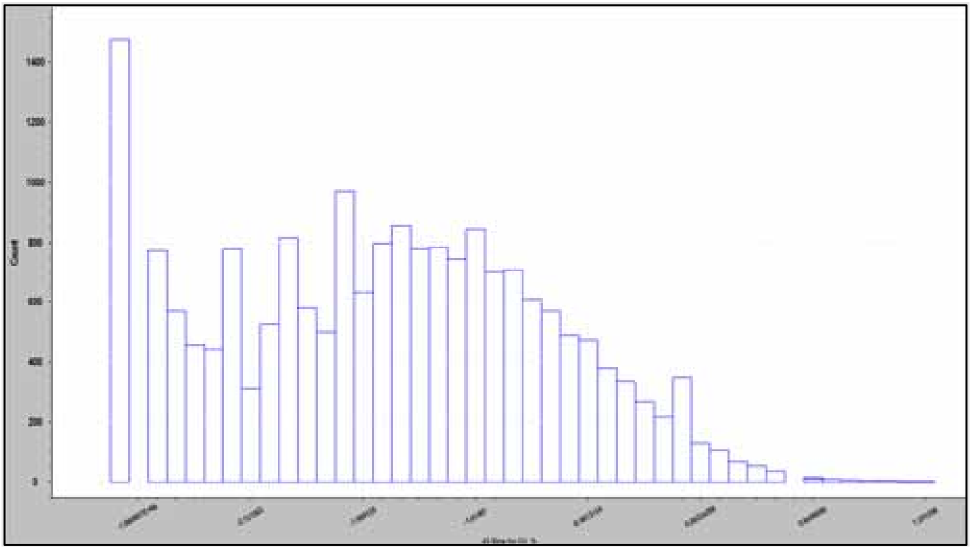


Figure 14-12: Log Histogram of Copper Assays from Lady Clayre Deposit

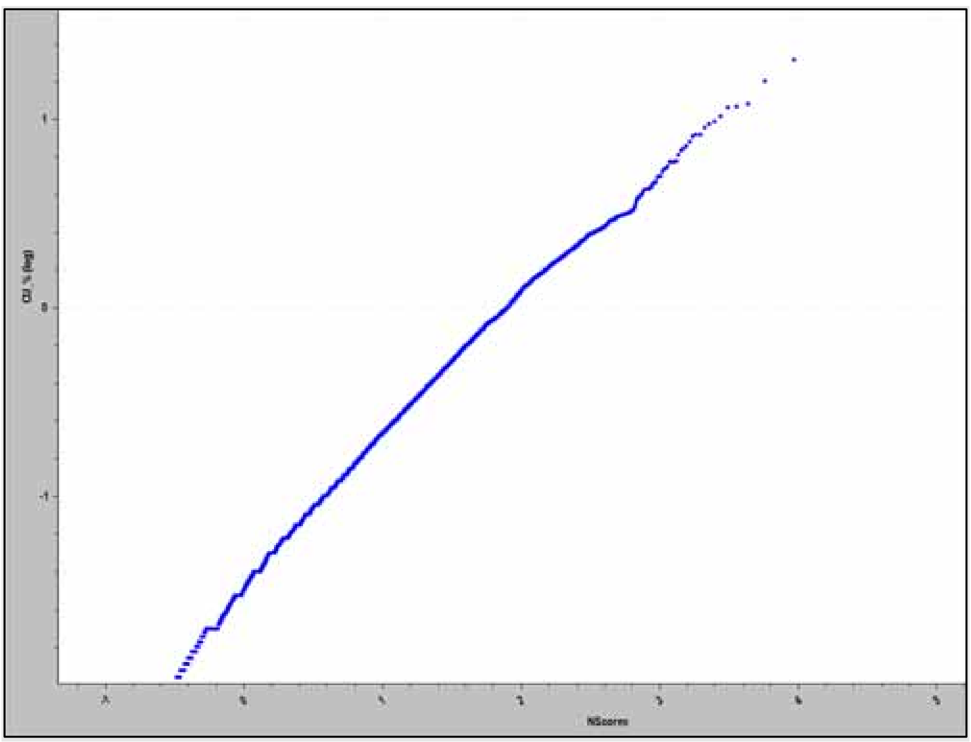


Figure 14-13: Log Probability Plot of Copper Assays from Lady Clayre Deposit

Table 14-15: Basic Statistics for 2.5 m Composites by Domain at Lady Clayre

	West Zone – Primary		East Zone – Primary		Oxide Zone – All	
	Cu	Au	Cu	Au	Cu	Au
Count	5,904	5,904	4,438	4,438	1,407	1,407
Mean	0.14	0.06	0.15	0.07	0.17	0.07
Median	0.044	0.01	0.04	0.01	0.06	0.01
Min.	0.00	0.00	0.00	0.00	0.00	0.00
Max.	17.00	10.43	7.37	10.52	3.6	10.5
Std. Dev.	0.42	0.29	0.36	0.36	0.32	0.36
CoV	2.89	4.94	2.37	5.31	1.91	5.30

Table 14-16: Basic Statistics for 2.5 m Composites by Domain at Ivy Ann

	Ivy Ann – Primary		Ivy Ann – Oxide		Ivy Ann North – Primary		Ivy Ann North – Oxide	
	Cu	Au	Cu	Au	Cu	Au	Cu	Au
Count	3,825	3,825	702	702	815	815	440	440
Mean	0.18	0.03	0.15	0.02	0.07	0.01	0.04	0.00
Median	0.04	0.00	0.04	0.00	0.02	0.00	0.01	0.00
Minimum	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum	4.00	0.30	2.58	0.35	2.85	0.50	0.5	0.2
Std. Dev.	0.35	0.05	0.28	0.04	0.18	0.03	0.06	0.02
CoV	1.97	1.95	1.86	1.92	2.53	3.54	1.62	3.93

14.5 Bulk Density

There are 1,862 bulk density measurements in the Little Eva deposit data base. The measurements were made on drill core using the “weigh in water, weigh in air” method. A histogram of the density measurements (Figure 14-11) indicates multiple populations: separating the data by rock type shows the separate populations. There were 1,386 bulk density measurements collected from rocks classified as the fresh volcanic package which has a mean bulk density of 2.80 t/m^3. A further 371 measurements were collected from the metasedimentary rocks, with a mean bulk density of 2.70 t/m^3. A total of 70 measurements were made on the felsic intrusive rocks, which yielded a mean bulk density of 2.63 t/m^3. These mean bulk density values were assigned to the blocks in the Little Eva block model.

The oxide zone did not have enough bulk density data points for detailed statistical analysis. A bulk density of 2.5 t/m^3 was assigned to all the oxide zone blocks in the block model. This value is based on the McDonald Speijers resource report from 2006.

Table 14-17: Bulk Density Data and Average or Assigned Values

Description	Bulk Density*	Sample Count
Overburden	1.5	0
Volcanics (Fresh)	2.8	1,386
Volcanics (Oxide)	2.5	18
Felsic Intrusive (Fresh)	2.63	70
Felsic Intrusive (Oxide)	2.5	0
Metasediments (Fresh)	2.7	371
Metasediments (Oxide)	2.5	17

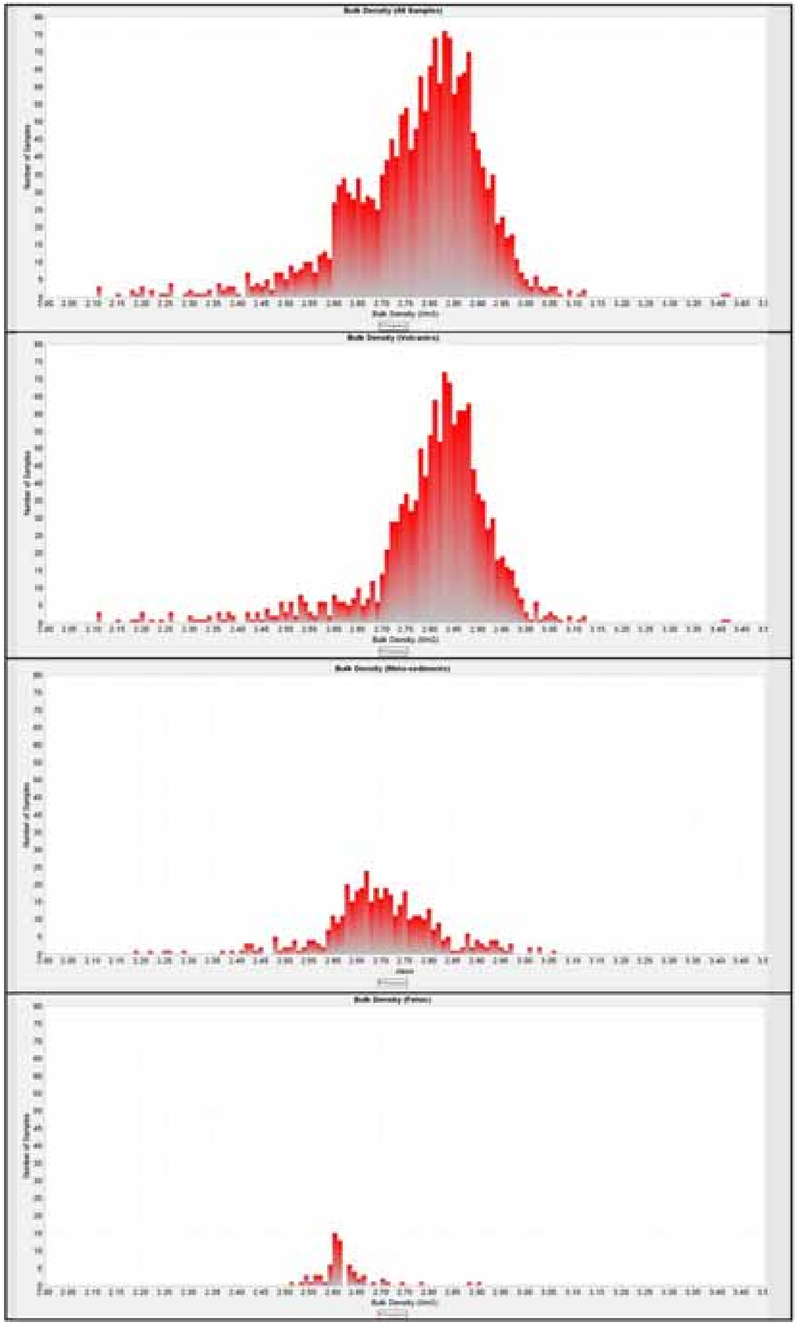
Limited bulk density measurements have been collected at Turkey Creek but are similar to the other zones and therefore a bulk density of 2.5 t/m^3 was assigned to the oxide material, and a bulk density of 2.7 t/m^3 was assigned to the fresh material. Bulk densities for the Blackard and Scanlan deposits were assigned to blocks based on their mineralogical domains, as shown in Table 14-18. The density determinations were derived from a limited, but relatively consistent, set of measurements completed during exploration and metallurgical testing.

SG measurements for the Ivy Ann, Bedford, and Lady Clayre deposits are limited. The oxide zones were assigned a value of 2.11 t/m^3, and for the sulphide zones and/or fresh rock an SG of 2.58 t/m^3 was assigned. Samples from the Bedford deposit suggest a higher density (2.78 t/m^3), which should be considered in future work.

Bulk densities for the Blackard and Scanlan deposits were assigned to blocks based on their mineralogical domains, as shown in Table 14-18. The density determinations were derived from over 618 historical data records (sourced from previous project operators) and the recent completion of 24 bulk sample tests on the Blackard deposit to confirm the historical findings.

Table 14-18: Bulk Density Used for Blackard and Scanlan Deposits

Description	Bulk Density
Oxide zone	2.08
Copper zone	2.18
Transition zone	2.35
Sulphide zone	2.50



Note: For all samples (top), volcanic rocks (2nd from top), metasediments (3rd from top), felsic intrusive (bottom)

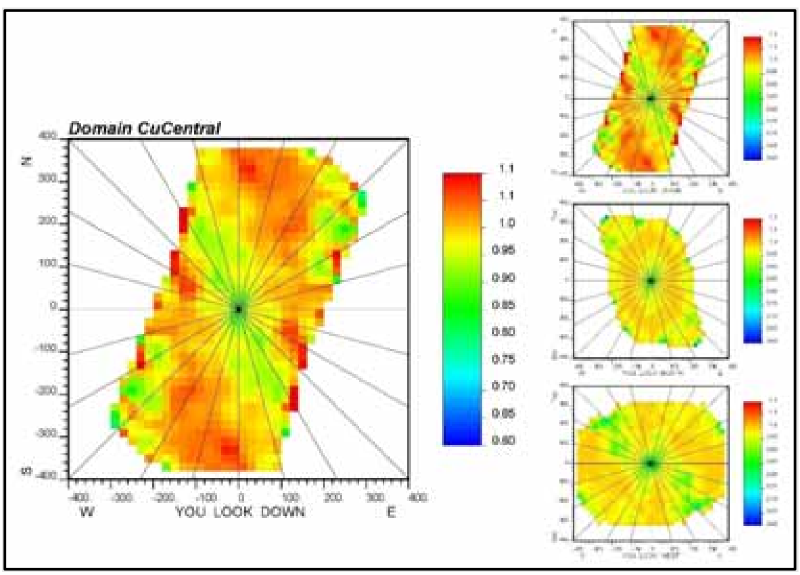
Figure 14-14: Bulk Density Histograms

14.6 Variography

A review of the Little Eva deposit variography was undertaken by SRK Vancouver and both downhole and directional correlograms were generated on logarithmic assay data within each of the four domains. Variography was only undertaken on the other deposits to investigate data continuity within the larger domains, but kriging interpolations were not used due to the limited size and shape of most of the domains and an ID^2 interpolation was used.

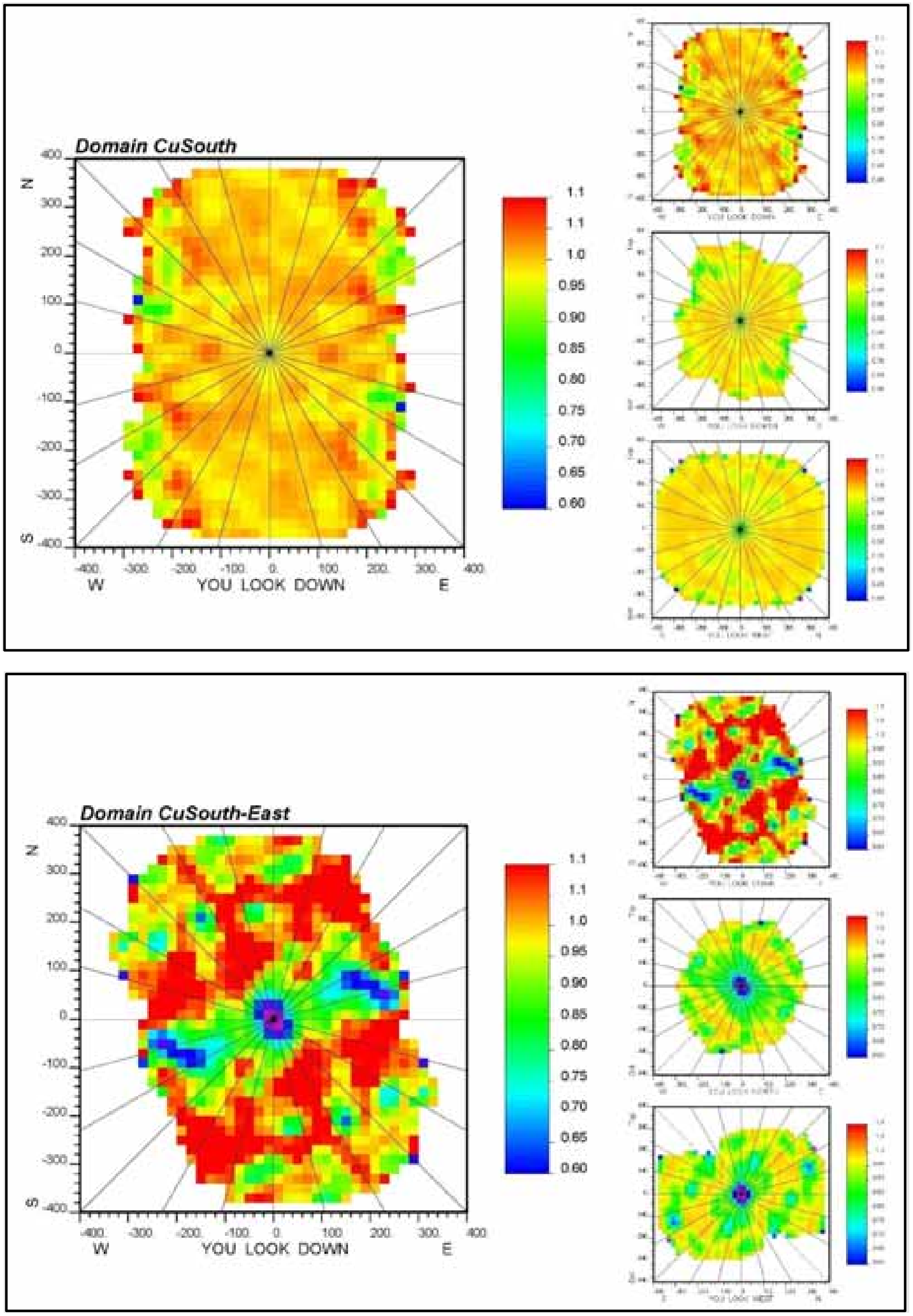
Variography is required to provide the necessary inputs to use the ordinary kriging (OK) method of interpolation. The semi-variogram is used to determine the spatial continuity of mineralization in 3D. The direction with the best continuity is referred to as the major axis, with the semi-major being the next-best direction of continuity, and the minor being the direction of least continuity in an orthogonal coordinate system. Semi-variograms provide measurements of three components of continuity: the nugget, the sill, and the range. The nugget is a measure of the randomness of samples, or put another way, the variability between samples over very short distances. There is an implicit assumption in grade modelling of mineral deposits that a spatial relationship exists between samples, and that this relationship is stronger between closely spaced samples but diminishes with increasing distance between samples. The sill is a measure of the point at which the maximum variability between samples is reached and this distance is referred to as the range. In addition to the variogram axes, nugget, sill, and range, the curve of the semi-variogram is modelled, and the type of model (e.g., spherical, exponential) is also used by the kriging program during interpolation.

3D analysis and the resulting semi-variograms were produced for copper in all the Little Eva domains. Geometric anisotropy was demonstrated in most cases, and nested exponential models were fitted to the data. Variogram maps, generated by the process of determining the orientations of maximum mineralization continuity and the appropriate lag distances, are displayed in Figure 14-15 and Figure 14-16. An example of a variogram (correlogram) is provided in Figure 14-17 and the parameters are summarized in the Table 14-19. Due to the high correlation between copper and gold, the same interpolation parameters were used for both elements.



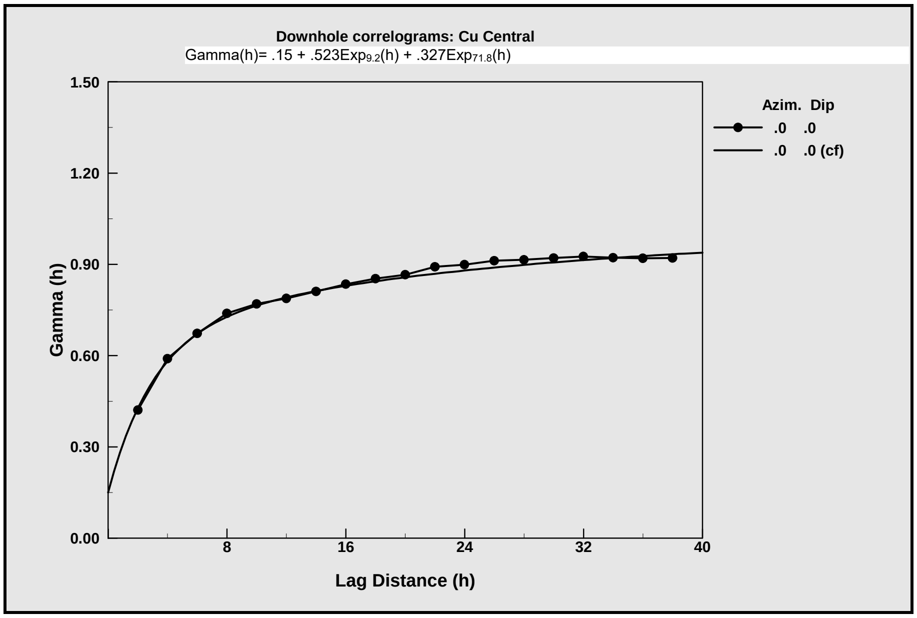
Note: North (upper) and Central (lower) domains. Large image is a plan view and the three images on the right side are plan, followed by cross and long sections. Cooler colours indicate directions of higher continuity (or lower grade variance). In the upper image the best continuity is displayed in the cross-section dipping to the east at 65 degrees. Continuity in the N-S orientation (plan view) reflects the trend of the zone. The lower image is from the Central zone which displays moderate continuity in a northwesterly direction but very little preferred orientation in cross-section and a faint southerly plunge in the long section.

Figure 14-15: Variogram for Little Eva



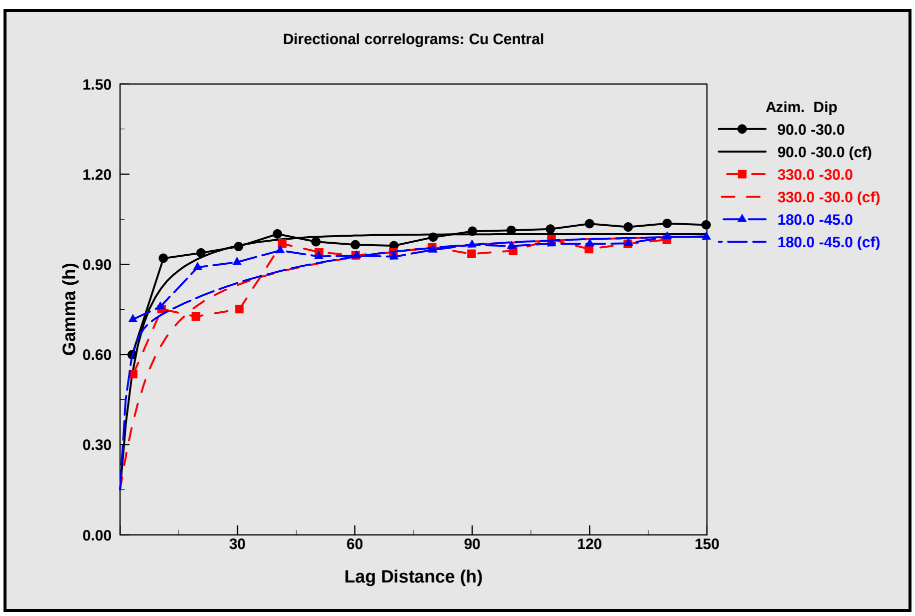
Note: The southern domain has no or very little indication of direction of continuity. The Southeast domain demonstrates a moderate north-easterly orientation in plan and weak dip to the east and weak northward plunge.

Figure 14-16: Variogram for South and Southeast Domains of the Little Eva Deposit



Note: Displays a Range of Approximately 25 m and Nugget Value of 0.15

Figure 14-17: Downhole Correlograms for the Central Domain



Note: Displays Limited Directional Preference, Relatively High Nugget Values, and a Range of around 45-50 m

Figure 14-18: Directional Correlograms for the Central Domain

Table 14-19: Summary of Variogram Parameters for Kriging Interpolation at Little Eva

Domain	Nugget (Co)	Principal Azimuth	Principal Dip	Intermediate Azimuth	Range 1 (m) – Exponential				Range 2 (m) – Exponential			
					Sill (C1)	X	Y	Z	Sill (C2)	X	Y	Z
North	0.2	98	31	360	0.5	6	25	25	0.3	50	100	500
Central	0.15	79	-23	333	0.5	10	20	5	0.35	40	120	130
South	0.3	90	-15	360	0.5	10	30	20	0.2	30	65	120

14.7 Grade Interpolation

For all domains, in all deposits grade interpolation was carried out with Gemcom software. Interpolation within the Little Eva deposit was carried out with ordinary OK for the North, Central and Southern domains and ID^2 for the South-east domain due to the reduced amount of data compared to the other domains. All other deposits were interpolated by ID^2 methods. Copper and gold (where appropriate) were interpolated within “3D solids models” that enclose the mineralized area below overburden. Both Little Eva and Turkey Creek used previously defined 3D models from Altona geologists. 3D shells to constrain interpolations for the Blackard and Scanlan deposits were created by CMMC and were designed to outline interpreted fold geometry and provide reasonable geological limits to linear interpolations. Interpolation of the Bedford deposit was constrained by an outer grade-shell, while the two southern deposits, Lady Clayre and Ivy Ann, were interpolated unconstrained due to poorly understood geometry of controls on mineralization.

Boundaries between domains in the Little Eva deposit were soft boundaries, where the search ellipse can use data across the boundary and the outer boundaries were hard, where data beyond the boundary is not used. There is an outer “low-data and low-grade” shell around the Central and South domains of the Little Eva deposit, and the scattered data within this shell is interpolated separately, as Inferred material with any blocks being generated added to the adjacent domain during resource tabulation.

For all blocks that were estimated by OK within the Little Eva deposit, the interpolation was conducted in a series of three passes with the dimensions and orientations of the search ellipsoid in each pass related to the semi-variogram parameters listed in Table 14-21. For the other blocks in Little Eva, and all the other deposits, which were interpolated with ID^2 methods, blocks were also estimated with three passes of increasing search size, corresponding to the Measured, Indicated, and Inferred classifications.

The search ellipsoid is defined by three orthogonal axis which are given lengths and orientations which reflects the interpreted continuity of mineralization (Figure 14-19). Thus, the shape of the search ellipse attempts to mimic the anisotropy of mineralization in each structural domain for each deposit. Orientation and dimensions of the searches are listed in tables for each deposit. The search orientations are given as the strike and plunge of the primary and secondary axis, the minor or tertiary axis is not required as it is perpendicular to the plane, which contains the primary and secondary axis. For a block to be estimated it must fit defined criteria of the search ellipse as listed in Table 14-20 and Table 14-21 for the Little Eva deposit and in subsequent tables for the other deposits. The search criteria include the dimensions of the search ellipse, and the minimum and maximum number of grade composites required collectively, as well as the minimum and maximum number of composites required from any drill hole. The three passes that determine the classification of the blocks are carried out with increasing dimensions of the search ellipse, where the maximum dimension is generally equal to, or less than the maximum ranges (continuity) indicated by the variograms. The different passes are usually geometric fractions of the variogram range. For the deposits interpolated with ID^2, variography from the larger, structurally linear sections of the deposit was used to provide support for search distances in smaller domains that were determined by a combination of visual inspection of grade distribution and drill spacing. Criteria for the interpolations within each deposit are listed in Table 14-22 through Table 14-31.

For blocks that did not fit the estimation criteria in the first pass, a second pass was completed with a larger search ellipse. A third and final pass was completed for any blocks that were not interpolated in the first two passes. If a block still is not interpolated by the third pass, then it is left blank. A maximum number of composites per drill hole is specified to ensure that an appropriate amount of data form adjacent drill holes is used. Since the composites used herein are ½ of the block size typically a maximum of 3 or 4 composites from any drill hole is set.

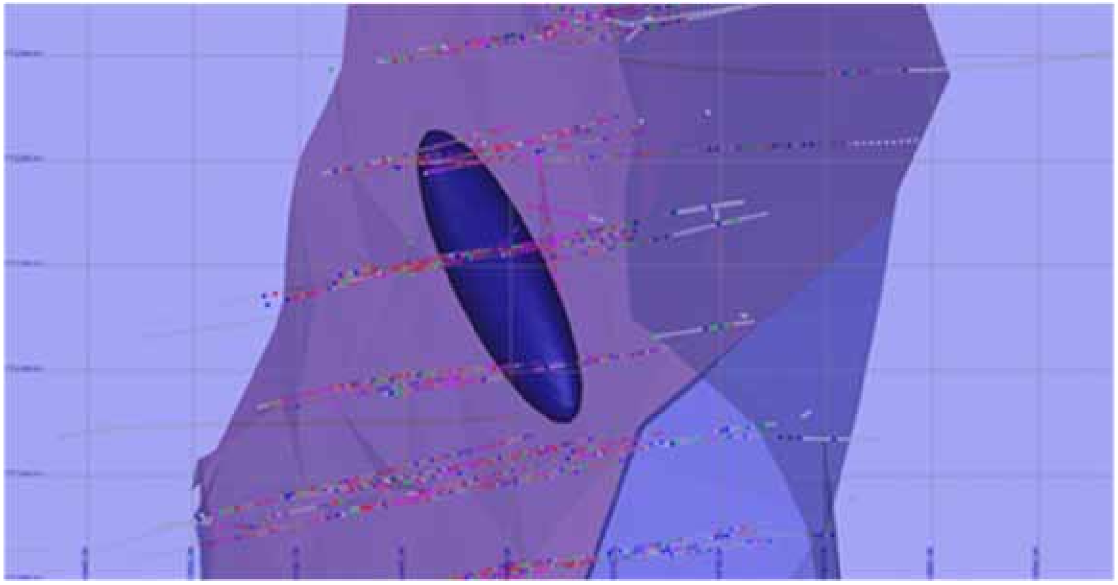
Table 14-20: Search Criteria for Interpolation for Little Eva

Pass	Criteria	Little Eva			
		North	Central	South	South-East
Measured	Minimum No. of Composites	5	5	5	5
	Maximum No. of Composites	9	9	9	9
	Maximum No. of Composites/Hole	2	2	2	2
Indicated	Minimum No. of Composites	7	5	4	5
	Maximum No. of Composites	10	9	10	12
	Maximum No. of Composites/Hole	2	2	3	2
Inferred	Minimum No. of Composites	3	4	3	4
	Maximum No. of Composites	9	12	8	12
	Maximum No. of Composites/Hole	2	3	3	3

Table 14-21: Search Ellipse for Interpolation for Little Eva

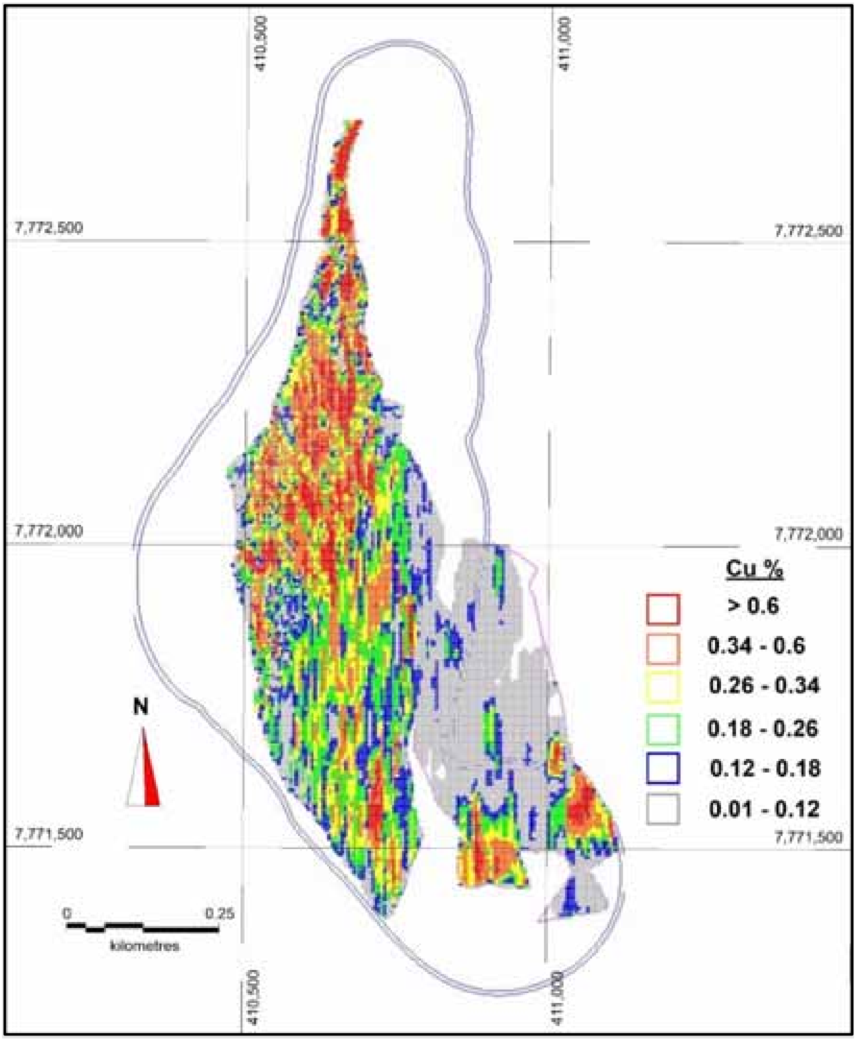
Domain	Pass	Class	X (m)	Y (m)	Z (m)	Principal Azimuth	Principal Plunge	Intermediate Azimuth	Intermediate Plunge
North	1	Measured	10	40	20				
10	2	Indicated	15	60	30	10	0	100	-50
	3	Inferred	20	75	40				
Central	1	Measured	10	60	20				
20	2	Indicated	20	90	30	333	0	243	-67
	3	Inferred	20	120	40				
South	1	Measured	15	50	40				
30	2	Indicated	25	75	55	0	0	270	-75
	3	Inferred	30	120	70				
South-East	1	Measured	40	60	15				
40	2	Indicated	50	90	20	0	0	270	-25
	3	Inferred	60	100	25				

Note: Azimuth and plunge direction given for principal and intermediate axis, right hand rule applies, dip is to right of azimuth, and the minor axis mutually perpendicular to the other two. This format is provided to help the reader visualize the search ellipse. Specifications used for the same ellipse in Gemcom are different.



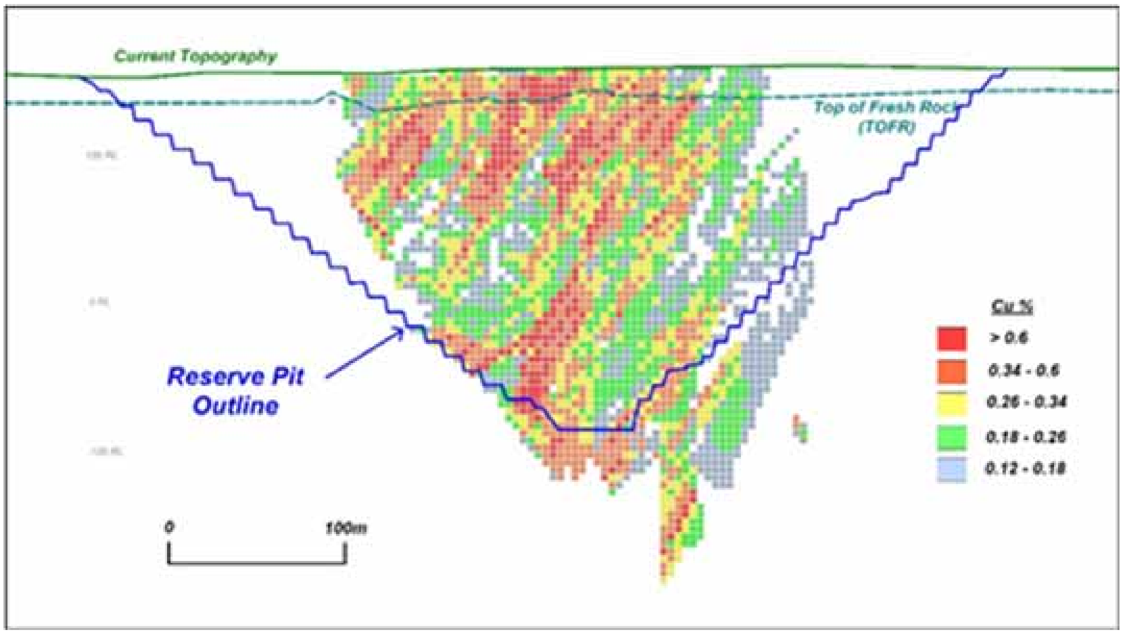
Note: Search ellipse has 60 m radius in long axis, a distance required to ensure data contribution from drill holes on at least two and sometimes on three sections. However, with the number of composites being specified (total minimum, total maximum, and a maximum number from a single hole), only the closest composites to the block centroid that meet the search criteria will be interpolated into the block).

Figure 14-19: View of the Central Domain (Little Eva), with Search Ellipse in Blue for Indicated Resources



Note: Two benches below the top of the sulphide zone.

Figure 14-20: Little Eva Deposit Plan View of Colour-Coded Block Grades at 120 m Elevation within Reserve Pit



Note: See above plan for location

Figure 14-21: Cross-Section 7,772,100N (see plan above) Displaying Colour-Coded Block Grades

At Turkey Creek, assay data above 0.01% Cu was examined and an inflection in the data distribution on the cumulative probability plot was noted at approximately 0.2% Cu. This confirms Altona's application of a 0.2% nominal cut-off grade for interpretation of a grade-shell outlining the copper mineralization.

CMMC used Altona's wireframe models of the copper mineralization to guide the interpretation of the copper domains at Turkey Creek. Mineralization within the Southern zone is generally tabular and is oriented north-south with dips at 60° to the east. At the northern end of the deposit, the strike of the mineralization swings sharply towards the east and dips steeply south: this zone is referred to as the Northern fold area. The mineralization within the Southern zone is truncated to the south and north by fault zones, (Figure 14-22). The mineralization within the Southern zone contains both hanging wall and footwall zones, with a narrow band of low-grade or waste between them; however, these structures were modelled together based on a 5 m^3 block size that should allow resolution between these zones. There is evidence of lower grade mineralization within the central part of the Northern fold area; however, drilling is too widely spaced to permit a robust interpretation of that horizon.

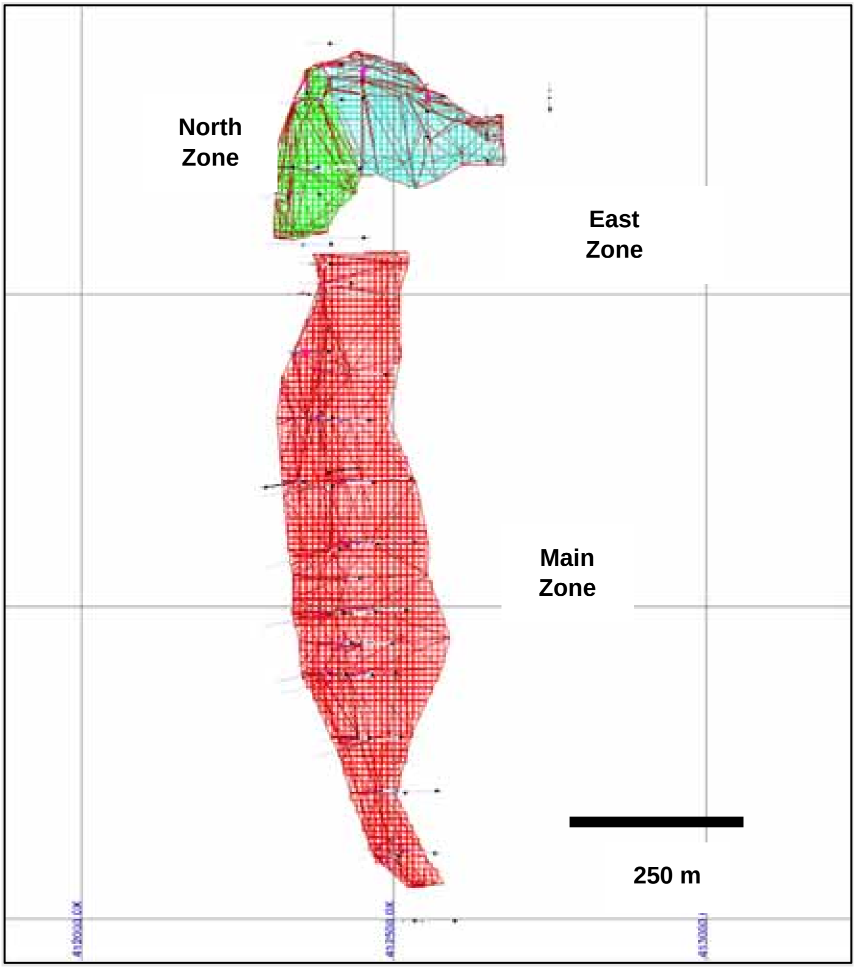


Figure 14-22: Wire framed Domains for the Turkey Creek Deposit

Table 14-22: Search Criteria for Interpolation for Turkey Creek

Pass	Criteria	Turkey Creek		
		South	North	North Fold
Measured	Minimum No. of Composites	3	3	3
	Maximum No. of Composites	12	12	12
	Maximum No. of Composites/Hole	2	2	2
Indicated	Minimum No. of Composites	3	3	3
	Maximum No. of Composites	12	12	12
	Maximum No. of Composites/Hole	2	2	2
Inferred	Minimum No. of Composites	3	3	3
	Maximum No. of Composites	12	12	12
	Maximum No. of Composites/Hole	2	2	2

Table 14-23: Search Ellipse Parameters by Domain for Turkey Creek

Domain	Pass	Class	X (m)	Y (m)	Z (m)	Principal Azimuth	Principal Plunge	Intermediate Azimuth	Intermediate Plunge
Main	1	Measured	20	50	50				
	2	Indicated	40	75	75	350	0	80	-65
	3	Inferred	60	100	100				
North	1	Measured	20	50	50				
	2	Indicated	40	75	75	20	0	110	-65
	3	Inferred	60	100	100				
East	1	Measured	20	50	50				
	2	Indicated	40	75	75	120	0	210	-65
	3	Inferred	60	100	100				

Note: see Table 24-21 for orientation information.

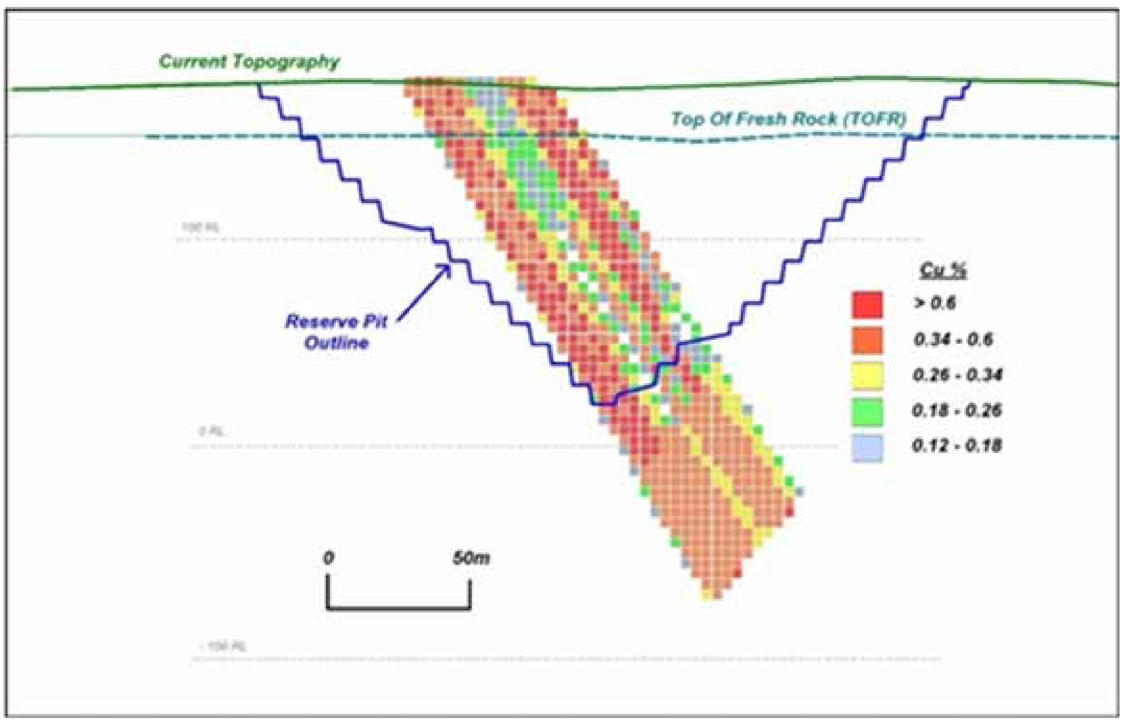
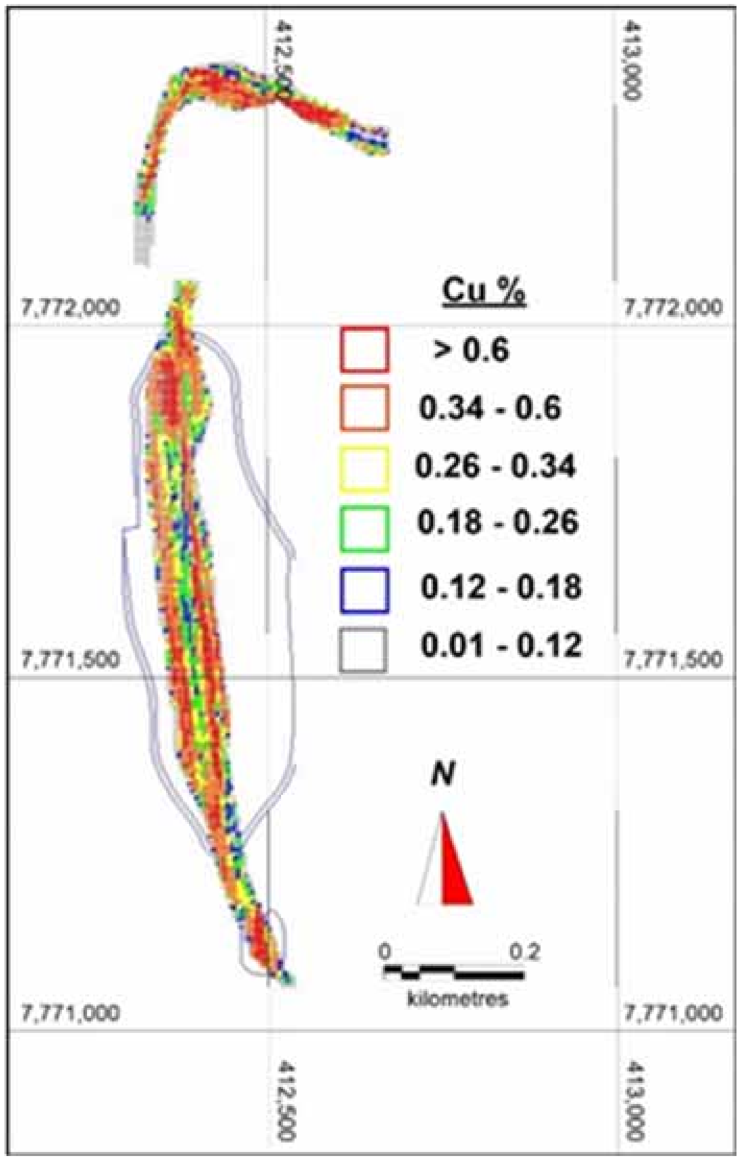


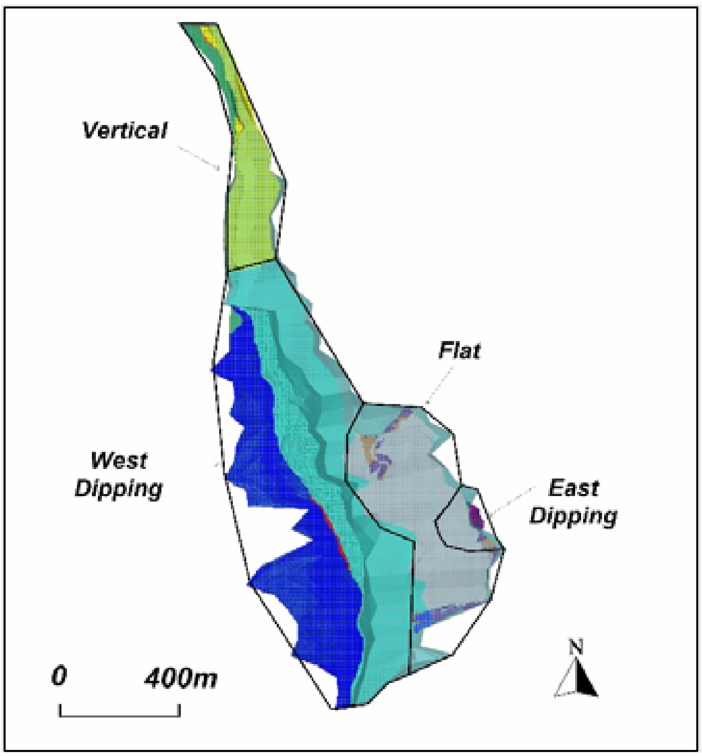
Figure 14-23: Turkey Creek Cross-Section at 7,771,500N (mid-point of Main Zone) of Colour-Coded Block Grades within Design Pit



Note: Two benches below the top of the sulphide zone.

Figure 14-24: Turkey Creek Plan View of Colour-Coded Block Grades at 120 m Elevation within Reserve Pit

The Blackard deposit was divided into three structural domains to reflect a west-dipping section, an adjacent flat to gently east-dipping section, and a northerly, steeply- to vertically-dipping section. It is recognized that these domains are an oversimplification of curved stratigraphy, but through the use of an outer constraining shell, a slightly larger interpolation search, and limiting the number of composites from a single drill hole, the block grades reliably reflect grade changes within the drill holes. The structural domains are illustrated in a plan view in Figure 14-25. Small changes in curvature, and dips along the deeper sections of the west-dipping fold limb were not completely captured but the volumes and grades would still be appropriately measured. Drill spacing at depth was insufficient to meet the distance requirements of the interpolation, even for the Inferred category, and therefore block grades were not estimated; however, the areas with insufficient drill density for estimation are generally will below the pit shell used to define Reserves. Specifying the number of grade composites used by the search ellipse during the interpolation is used both to ensure a requisite number of drill holes are used for a particular classification, and also to limit the composites used to preserve sharp grade changes. For the Blackard interpolation, the maximum number of composites used in any search was 16, the maximum number used per drill hole was 3, and the minimum numbers of total composites required were 7, 5, and 4 for the Measured, Indicated, and Inferred classifications, respectively. This means that the search ellipse would use data from a minimum of 3 holes and a maximum of 5 holes for the Measured class, and a minimum of 2 holes for both the Indicated and Inferred classes, although the Indicated class would require more composites than the Inferred class. The requirement of data from at least two drill holes for the Inferred classification was used to prevent large clusters of data around single, isolated drill holes that would result from the relatively large searches required to cover two or three drill sections, but does result in some alternating bands of classification in areas where the deposit trend deviates from the primary search axis, however this has limited impact on the overall resource classification or reserve values.

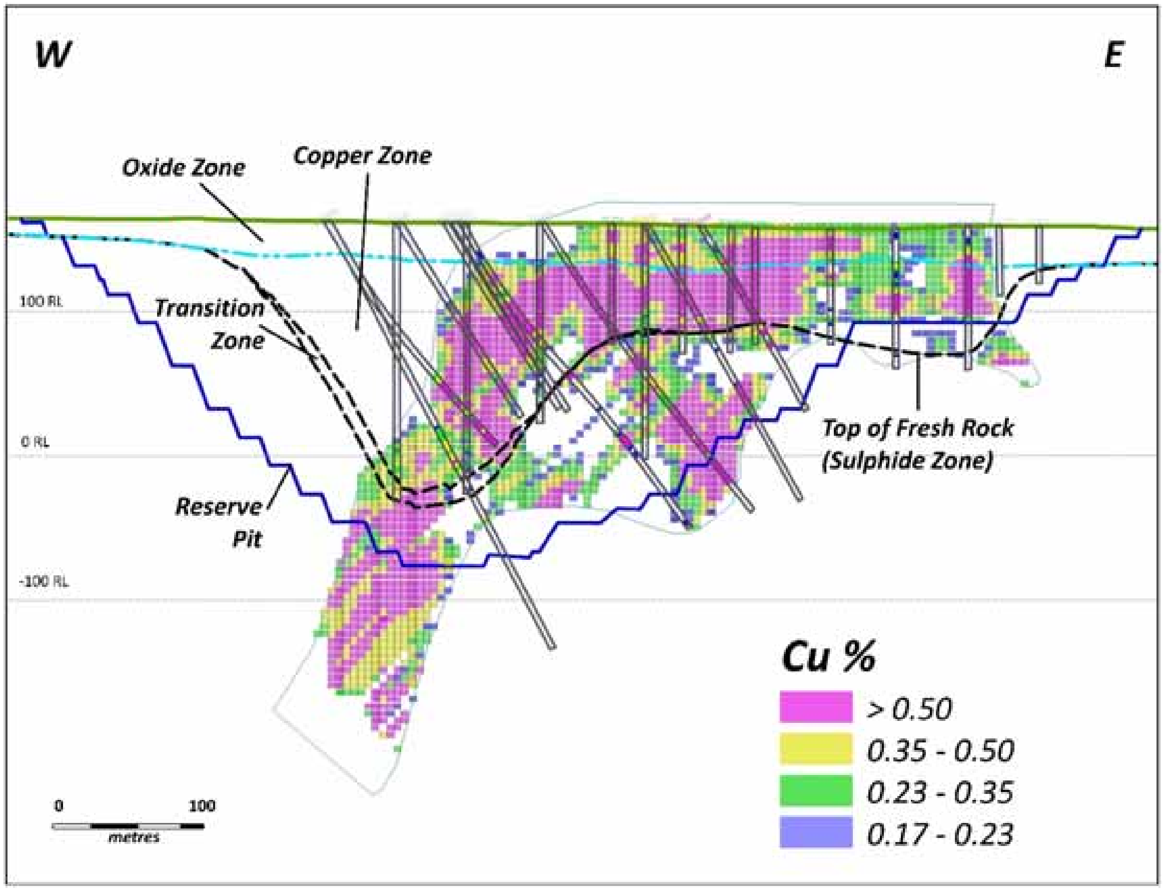


Note: Copper mineralogy domains are displayed in Figure 14-20

Figure 14-25: Structural Resource Estimation Domains of the Blackard Deposit in Plan

Table 14-24: Interpolation Parameters for the Blackard Deposit

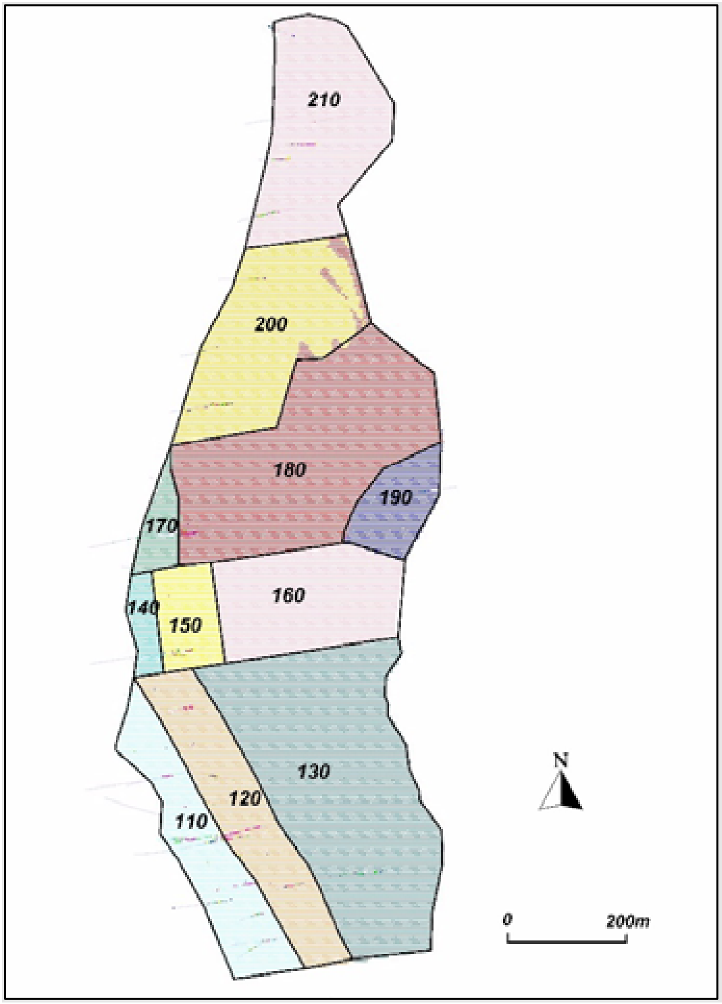
Structural Domain	Interpolation Parameters						Pass		
	Cu Domain	Search Orientation				Distance	1 Measured	2 Indicated	3 Inferred
		1st Azimuth	Plunge	2nd Azimuth	Plunge				
West Dipping						X (m)	25	50	120
	20					Y (m)	15	25	35
						Z (m)	60	90	120
						X (m)	25	50	120
	30	180	-10	270	-56	Y (m)	15	25	35
						Z (m)	60	90	120
						X (m)	25	50	120
	40					Y (m)	15	25	35
						Z (m)	60	90	120
Flat						X (m)	25	50	90
	25					Y (m)	15	25	35
						Z (m)	60	90	120
						X (m)	25	50	90
	35	0	0	270	0	Y (m)	15	25	35
						Z (m)	60	90	120
						X (m)	25	50	90
	45					Y (m)	15	25	35
						Z (m)	60	90	120
Vertical (North)						X (m)	25	50	90
	50					Y (m)	15	25	35
						Z (m)	60	90	120
						X (m)	25	50	90
	60	0	0	270	-90	Y (m)	15	25	35
						Z (m)	60	90	120
						X (m)	25	50	90
	70					Y (m)	15	25	35
						Z (m)	60	90	120



Notes: Colour-coded block grades within model can be compared to drill hole grades (bold). The US$2.75/lb Cu reserve design pit is shown in blue. Note that where high-grade (or low-grade) blocks do not show continuity with adjacent drill holes in the section, there must be continuity along strike (adjacent sections) for a grade to be interpolated.

Figure 14-26: Blackard Deposit Cross-Section at 7,765,150N

The Scanlan deposit is folded so that there are generally three different dip directions of mineralization across the deposit, which together with some fault offsets and changes in the deposit strike orientation results in multiple domains as illustrated in Figure 14-27. The smaller Bedford, Lady Clayre, and Ivy Ann deposits were estimated with just 2 or 3 domains.



Note: Each domain has differing mineralization trends which required corresponding search orientations for block grade interpolation. Details of the interpolation ellipse orientation are provided in Table 14-25.

Figure 14-27: Plan View of the Scanlan Deposit with Structural Domains Displayed

Table 14-25: Search Criteria for Interpolation for the Scanlan Deposit

Structural Domains	Class	X (m)	Y (m)	Z (m)	Mini. No. Comp.	Max. No. Comp.	Max. No. Comp./DH	1st Azimuth	Plunge	2nd Azimuth	Plunge
110	Measured	40	5	70	9	16	4	330	0	60	-60
	Indicated	60	7	80	7	16	4				
	Inferred	80	10	90	5	12	4				
120	Measured	40	5	70	9	16	4	330	0	240	-60
	Indicated	60	7	80	7	16	4				
	Inferred	80	10	90	5	12	4				
130	Measured	45	6	80	9	16	4	330	0	240	0
	Indicated	60	8	100	7	16	4				
	Inferred	75	12	120	5	12	4				
140	Measured	40	5	70	9	16	4	355	0	265	0
	Indicated	60	7	80	7	16	4				
	Inferred	80	10	90	5	12	4				
150	Measured	50	6	80	9	16	4	355	0	265	-60
	Indicated	70	8	100	7	16	4				
	Inferred	90	12	120	5	12	4				
160	Measured	45	5	80	9	16	4	355	0	265	0
	Indicated	65	7	90	7	16	4				
	Inferred	85	10	100	5	12	4				
170	Measured	50	6	80	9	16	4	12	0	282	-60
	Indicated	60	8	90	7	16	4				
	Inferred	80	10	100	5	12	4				
180	Measured	40	5	70	9	16	4	12	0	282	0
	Indicated	60	7	80	7	16	4				
	Inferred	80	10	90	5	12	4				
190	Measured	45	6	80	9	16	4	12	0	102	-40
	Indicated	70	8	100	7	16	4				
	Inferred	95	12	120	5	12	4				
200	Measured	40	5	70	9	16	4	12	0	282	-55
	Indicated	60	7	80	7	16	4				
	Inferred	80	10	90	5	12	4				
210	Measured	45	6	80	9	16	4	12	0	102	-70
	Indicated	70	8	90	7	16	4				
	Inferred	90	12	120	5	12	4				

Note: see Table 24-21 for orientation information.

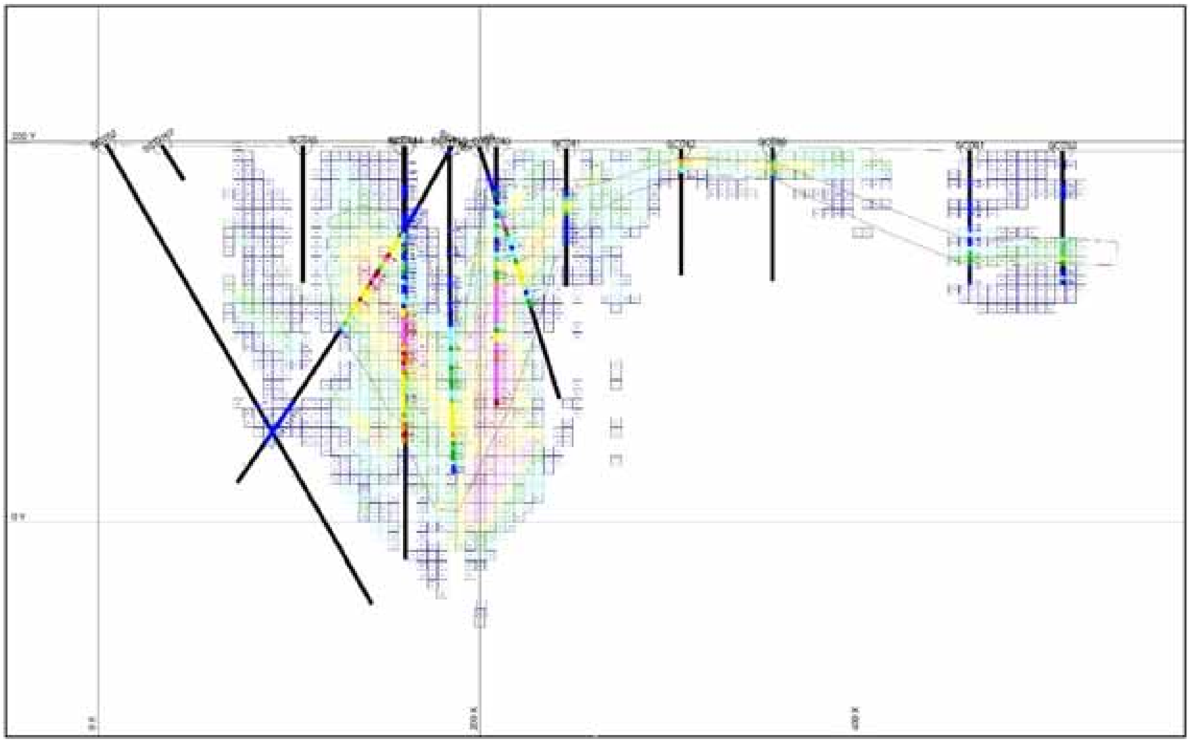


Figure 14-28: Cross-Section of Scanlan Deposit with Drill Hole, Block Grades and Resource Constraining Shell Displayed

Table 14-26: Search Criteria for Interpolation for the Bedford Deposit

Pass	Criteria	Bedford	
		North	South
Indicated	Minimum No. of Composites	4	4
	Maximum No. of Composites	12	12
	Maximum No of Composites/Hole	3	3
Inferred	Minimum No. of Composites	4	4
	Maximum No. of Composites	12	12
	Maximum No. of Composites/Hole	3	3

Table 14-27: Search Ellipse Parameters by Domain for the Bedford Deposit

Domain	Pass	Class	X (m)	Y (m)	Z (m)	1st Azimuth	Plunge	2nd Azimuth	Plunge
North	1	Indicated	9	40	20				
	2	Inferred	15	60	30	0	0	270	-70
South	1	Indicated	9	40	20				
	2	Inferred	15	60	30				

Note: *Gems Search anisotropy: Azimuth, Dip, Azimuth

Table 14-28: Search Criteria for Interpolation for the Lady Clayre Deposit

Pass	Criteria	Lady Clayre	
		East	West
Measured	Minimum No. of Composites	5	5
	Maximum No. of Composites	15	15
	Maximum No. of Composites/Hole	3	3
Indicated	Minimum No. of Composites	5	5
	Maximum No. of Composites	15	15
	Maximum No. of Composites/Hole	3	3
Inferred	Minimum No. of Composites	5	5
	Maximum No. of Composites	15	15
	Maximum No. of Composites/Hole	3	3

Table 14-29: Search Ellipse Parameters by Domain for the Lady Clayre Deposit

Domain	Pass	Class	X (m)	Y (m)	Z (m)	1st Azimuth	Plunge	2nd Azimuth	Plunge
East (77)	1	Measured	25	30	10				
	2	Indicated	30	37.5	12.5	35	0	305	-45
	3	Inferred	50	60	25				
West (66)	1	Measured	25	30	10				
	2	Indicated	30	37.5	12.5	345	0	255	-50
	3	Inferred	50	60	25				

Note: see Table 24-21 for orientation information

Table 14-30: Search Criteria for Interpolation for the Ivy Ann Deposit

Pass	Criteria	Ivy Ann	
		Ivy Ann	Ivy Ann North
Measured	Minimum No. of Composites	5	5
	Maximum No. of Composites	15	15
	Maximum No. of Composites/Hole	3	3
Indicated	Minimum No. of Composites	5	5
	Maximum No. of Composites	15	15
	Maximum No. of Composites/Hole	3	3
Inferred	Minimum No. of Composites	5	5
	Maximum No. of Composites	15	15
	Maximum No. of Composites/Hole	3	3

Table 14-31: Search Ellipse Parameters by Domain for the Ivy Ann Deposit

Domain	Pass	Class	X (m)	Y (m)	Z (m)	Principal Azimuth	Principal Plunge	Intermediate Azimuth	Intermediate Plunge
Ivy Ann South	1	Measured	20	25	5	26	0	116	-46
	2	Indicated	40	45	20				
	3	Inferred	70	80	40				
Ivy Ann North	1	Measured	20	25	5	35	0	125	-80
	2	Indicated	40	45	20				
	3	Inferred	70	80	40				

Note: *Gems Search anisotropy: Azimuth, Dip, Azimuth

14.8 Classification and Mineral Resource Statement

Estimated blocks within the different deposit models were tabulated between an upper and lower surface. For the sulphide part of the deposits, the upper surface was the base of the oxide or top of the sulphide zone boundary, and the lower surface was the constraining Whittle pit shell. The constraining pit shell is generated using the same operating and processing costs listed in Section 15, but with a $3.50/lb Cu price and $1,250/oz Au price used to limit Inferred resources to those having reasonable prospects of extraction. For the copper-only deposits the blocks were tabulated between surfaces defined by the base of the oxide zone, base of copper zone, base of transition zone, and the constraining resource pit.

Classification of the resources is based on definitions from CIM (2020) in accordance with NI 43-101 regulations. Resources are reported by deposit type and classification in Table 14-32 and then Measured and Indicated resources for each deposit are reported at three cut-off grades in Table 14-33.

Table 14-32: Copper Cut-off Grades (% based on NSR values) for Variable Recovery Mineralogical Zones in Blackard and Scanlan Deposits

Mineral Zone	Blackard Deposit			Scanlan Deposit		
Cut-off	Low	Mid	High	Low	Mid	High
Copper zone	0.24	0.32	0.39	0.26	0.32	0.39
Transition zone	0.20	0.27	0.33	0.20	0.27	0.33
Sulphide zone	0.17	0.23	0.28	0.17	0.23	0.28

Table 14-33: Eva Copper Project Resources by Category and Deposit at 0.17% Cu Cut-off Grade

	Tonnes (kt)	Cu Grade (% Cu)	Au Grade (g/t)	Cu Pounds (Mlb)	Au Ounces (koz)
Measured					
Little Eva	56,671	0.39	0.07	492	129
Turkey Creek	6,938	0.47	-	72	-
Blackard*	30,595	0.51	-	343	-
Scanlan*	11,397	0.59	-	147	-
Bedford	-	-			
Lady Clayre	5,113	0.42	0.17	47	28
Ivy Ann	1,107	0.38	0.07	9	3
Total Measured	**111,821**	**0.45**	**0.05**	**1,110**	**160**
Indicated					
Little Eva	65,154	0.34	0.07	486	135
Turkey Creek	6,871	0.44	-	67	-
Blackard*	53,073	0.45	-	521	=
Scanlan*	14,453	0.46	-	146	-
Bedford	3,002	0.54	0.14	36	14
Lady Clayre	2,228	0.40	0.18	20	13
Ivy Ann	4,037	0.35	0.08	31	10
Total Indicated	**148,818**	**0.40**	**0.04**	**1,310**	**172**
Measured+Indicated					
Little Eva	121,826	0.36	0.07	978	264
Turkey Creek	13,808	0.46	-	140	-
Blackard*	83,688	0.47	-	864	
Scanlan*	25,850	0.52	-	294	-
Bedford	3,002	0.54	0.14	36	14
Lady Clayre	7,341	0.41	0.17	66	40
Ivy Ann	5,144	0.36	0.08	41	13
Total Measured+Indicated	**260,659**	**0.42**	**0.04**	**2,419**	**330**
Inferred					
Little Eva	3,764	0.31	0.07	75	23
Turkey Creek	12,897	0.40	-	46	-
Blackard*	19,457	0.48	-	207	-
Scanlan*	3,432	0.44	-	33	-
Bedford	792	0.42	0.14	7	3
Lady Clayre	4,964	0.36	0.15	40	23
Ivy Ann	961	0.32	0.07	7	2
Total Inferred	**46,267**	**0.42**	**0.04**	**428**	**51**

Note: Mineral Resources

1. Joint Ore Reserves Code (JORC) and CIM definitions were followed for Mineral Resources.
2. Mineral Resources are inclusive of Mineral Reserves.
3. Mineral Resources are constrained within a Whittle pit shell generated with a copper price of $3.50/lb, a gold price of $1,250/oz and an exchange rate of AU$1.35 = US$1.00.
4. Density measurements were applied
5. Significant figures have been reduced to reflect uncertainty of estimations and therefore numbers may not add due to rounding.

Table 14-34: Mineral Resources for Eva Copper Project (inclusive of Reserves) at Different Cu Cut-off Grades

Deposit	Classification	Cut-off Grade	Tonnes (kt)	Grades		Contained Metal	
				Cu (%)	Au (g/t)	Cu Pounds (M lb)	Au Ounces (koz)
Little Eva	Measured+Indicated	0.30	58,047	0.51	0.09	656	168
		0.24	87,724	0.44	0.08	802	226
		0.17	121,826	0.36	0.07	977	274
	Inferred	0.17	3,764	0.31	0.07	26	23
Turkey Creek	Measured+Indicated	0.30	10,233	0.54	-	122	-
		0.24	12,036	0.50	-	133	-
		0.17	13,808	0.46	-	140	-
	Inferred	0.17	12,897	0.40	-	113	-
Blackard	Measured+Indicated	HG	52,492	0.60	-	690	-
		MG	62,677	0.55	-	758	-
		LG	77,320	0.49	-	836	-
	Inferred	LG	19,304	0.49	-	209	-
Scanlan	Measured+Indicated	HG	15,712	0.67	-	232	-
		MG	19,027	0.61	-	256	-
		LG	21,695	0.57	-	273	-
	Inferred	LG	2,671	0.50	-	29	-
Bedford	Measured+Indicated	0.30	2,010	0.69	0.18	30	12
		0.24	2,425	0.62	0.16	33	13
		0.17	3,002	0.54	0.14	36	14
	Inferred	0.17	792	0.42	0.14	7	3
Lady Clayre	Measured+Indicated	0.30	3,912	0.57	0.25	49	31
		0.24	5,198	0.50	0.21	57	35
		0.17	7,341	0.41	0.17	66	41
	Inferred	0.17	4,964	0.36	0.15	40	23
Ivy Ann	Measured+Indicated	0.30	2,980	0.45	0.09	30	9
		0.24	3,890	0.41	0.09	35	11
		0.17	5,144	0.36	0.08	41	12
	Inferred	0.17	961	0.32	0.07	7	2
Total	**Measured+Indicated**	**0.30**	**145,386**	**0.56**	**0.05**	**1,808**	**220**
		0.24	192,977	0.49	0.05	2,075	285
		0.17	250,136	0.43	0.04	2,369	341
Total	**Inferred**	**0.17**	**45,353**	**0.42**	**0.04**	**431**	**51**

Notes: Blackard and Scanlan deposit cut-off grades are based on NSR values which vary by weathering profile to reflect estimated recoveries and distance from the processing plant. Copper cut-off grades for the low-, mid-, and high-grade cut-offs are provided in Table 14-32.

<u>Mineral Resources:</u>

1. Joint Ore Reserves Code (JORC) and CIM definitions were followed for Mineral Resources.
2. Mineral Resources are inclusive of Mineral Reserves.
3. Mineral Resources are constrained within a Whittle pit shell generated with a copper price of \$3.50/lb, a gold price of \$1,250/oz and an exchange rate of AU\$1.35 = US\$1.00.
4. Density measurements were applied (ranges from 2.4 t/m^3 to 3.00 t/m^3).
5. Significant figures have been reduced to reflect uncertainty of estimations and therefore numbers may not add due to rounding.
LG = low grade; MG = medium grade; HG = high grade.

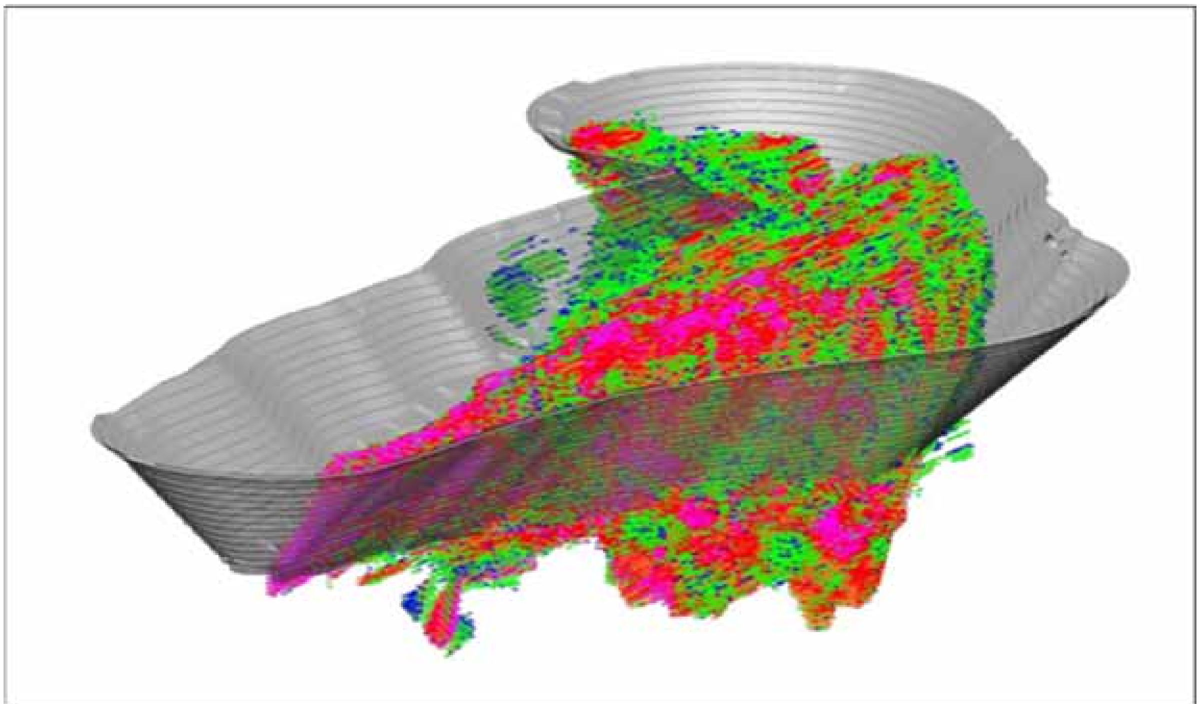
The three cut-off grades are used to separate waste material from expected low-grade, mid-grade and high-grade mill feed to allow the mine to maximize NPV using a stock-piling strategy. Oxide material overlies all the deposits and carries potentially economic copper grades and was estimated at the same time and with the same methods used for the sulphide material. Oxide resources were tabulated between the bottom of the oxide zone and topographic surface. At present, there is no demonstrated process to economically recover copper from the oxide zones; however, as this material will be removed by mining, it would be possible to stockpile the oxide material for potential processing at some future date. The oxide resources are presented by deposit and classification in Table 14-35.

Table 14-35: Oxide Mineral Resources for the Eva Copper Project

Deposit	Tonnes (kt)	Cu (%)	Au (g/t)	Cu Pound (Mlb)	Au Ounces (koz)
Measured+Indicated					
Little Eva	7,108	0.36	0.08	56	18
Turkey Creek	3,398	0.47	0	53	0
Bedford	775	0.61	0.15	10	4
Lady Clayre	1,598	0.35	0.12	12	6
Ivy Ann	1,195	0.31	0.06	8	2
Inferred					
Little Eva	31	0.41	0.03	0.3	0
Turkey Creek	1,301	0.51	0	15	0
Bedford	384	0.49	0.15	4	2
Lady Clayre	911	0.27	0.09	6	3
Ivy Ann	371	0.26	0.05	2	1
Total Measured+Indicated	**14,074**	**0.40**	**0.07**	**123**	**30**
Total Inferred	**2,998**	**0.40**	**0.05**	**26**	**5**

Notes: Mineral Resource:
1. JORC and CIM definitions were followed for Mineral Resources.
2. Oxide Mineral Resources are constrained within a Whittle pit shell as described in the text.
3. Density value of 2.4 t/m^3 was applied to all oxide zones.
4. Significant figures have been reduced to reflect uncertainty of estimations and therefore numbers may not add due to rounding.
5. Oxide material hosted within carbonate rich rocks overlying Blackard and Scanlan deposits are not included.

The Eva Project hosts additional copper-only deposits that have received exploration attention in the past for which historical resource estimates exist as listed in Table 14-36. These copper-only deposits are similar to the Blackard and Scanlan deposits, hosted within the same stratigraphy and with the same deep weathering profiles, containing a mix of copper oxide minerals, native copper and other copper bearing minerals, transitioning to sulphide minerals at depth. Assuming the same processing method that is planned for use with the Blackard and Scanlan deposits, these deposits should be considered for further exploration, in particular the Legend deposit, which is the northern extension of the Blackard system and is proximal to the proposed processing plant.



Note: Legend for block colours are the same as in Figure 14-32. Note that isometric view makes blocks appear lower towards the south because of a gap between the pit and block edge, sulphide resource blocks generally occur between the 2nd and 3rd benches.

Figure 14-29: Isometric View (looking south) of the Resource Block Model at 0.17% Cut-off at Top of Sulphide Zone, within Reserve Pit

Table 14-36: Historical Resource Estimates for Copper-Only Deposit Mineral Resources

Deposits	Tonnes (Mt)	Cu (%)	Cu Pounds (Mlb)
Legend	17.4	0.54	207
Great Southern	6.0	0.61	81
Longamundi	10.4	0.66	151
Caroline	3.6	0.53	42
Charlie Brown	0.7	0.40	6
Total	**38.1**	**0.58**	**487**

Notes: Mineral Resource:

1. Historical Resources should not be relied upon.

2. Significant figures have been reduced to reflect uncertainty of estimations and therefore numbers may not add due to rounding.

14.9 Resource Verification

The resource block models were examined for validity and reasonableness by several methods:

- Visual comparison of block grades relative to drill holes on cross-sections
- Comparison of statistical summary of assay, composite, and block grades
- Comparison of different method of interpolations such as OK to ID^2 or Nearest Neighbour (NN)
- Comparison to past estimations

A basic method of validation is to compare drill hole composites to adjacent block grades on plans and sections. While this method demonstrates that block grades are reasonable and accurately reflect drill data, since block grades are interpolated from data at some distance from the section, it does not necessarily follow that the block grade will exactly match the proximal drill hole. Additionally, it is not possible to examine every block value, thus this method may only reveal significant problems with an interpolation. Examination of drill hole grades relative to adjacent block grades demonstrates a good degree of correspondence, suggesting that block grades are fairly representing drill hole composites, as illustrated in Figure 14-30 through Figure 14-34.

Drill hole LED1015 was drilled by CMMC in 2018 to obtain material within the starter pit for metallurgical testing. The core was shipped to the Copper Mountain Mine site where it was logged, split, and assayed. LED1015 was not used in the resource estimation. Drill hole composites are plotted along with block grades on a cross-section in Figure 14-31 and are well correlated, indicating that the interpolation is working well in this location. Similarly, LED1006 was drilled by SRIG in 2015 and quarter-core was shipped to the Copper Mountain Mine for assaying and metallurgical testing. The two SRIG drill holes were inadvertently left out of the resource database and were not used for resource estimation. The drill hole composites compare well, but not perfectly, with the interpolated blocks (Figure 14-30 and Figure 14-31) indicating a reasonableness of the interpolation. Comparison of mean copper grades from drill holes, composites, and Measured and Indicated blocks by domain from the Little Eva deposit are displayed graphically in Figure 14-33 and Figure 14-34. As would be expected, the mean of the 2.5 m composite grades is slightly lower than the mean grades from the 1 m samples, whereas the 5 m cubic blocks are lower again. The difference between the mean grade of assays and the mean grades of the blocks is a function of volume variance and indicates the incorporation of lower-grade or barren material as the sample volume is increased. Since data used to estimate block grades is taken from numerous composites and drill holes within the search ellipse, it is expected that some low-grade or barren material will be incorporated as dilution, a feature that becomes more accentuated by selecting data above the copper cut-off grade (Figure 14-35).

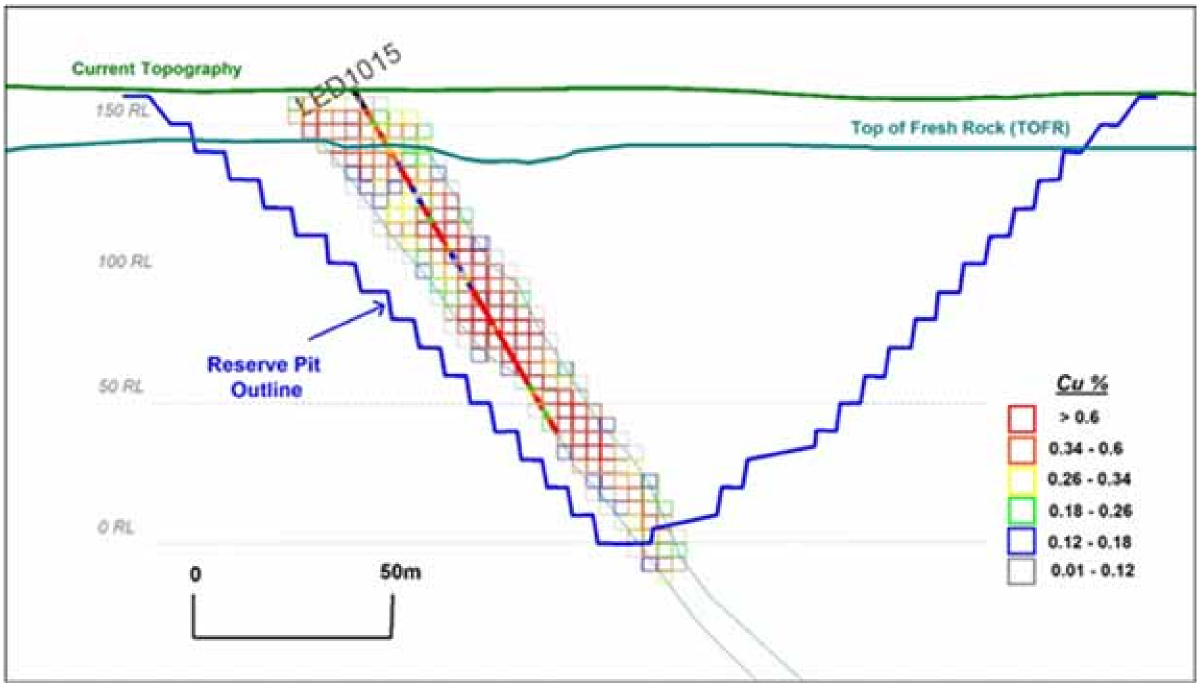


Figure 14-30: Cross-Section through the North End of the Eva Deposit with Block Grades and Drill Hole Composites from LED1015 Drilled Post Estimation

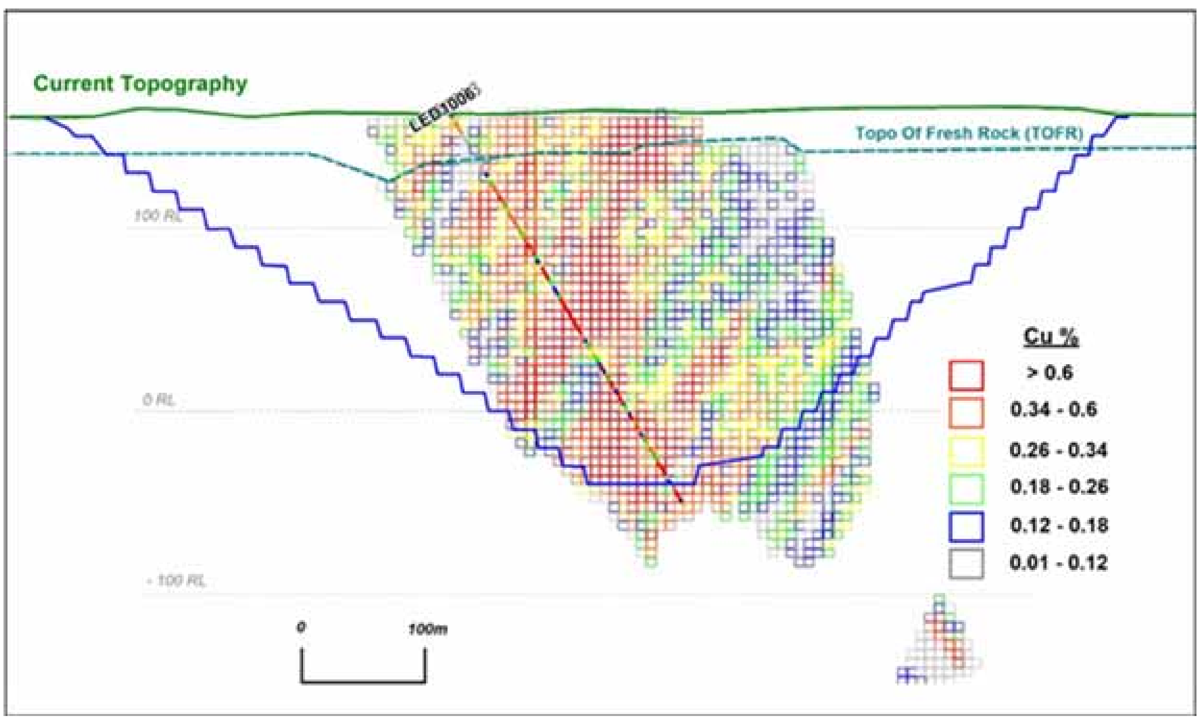
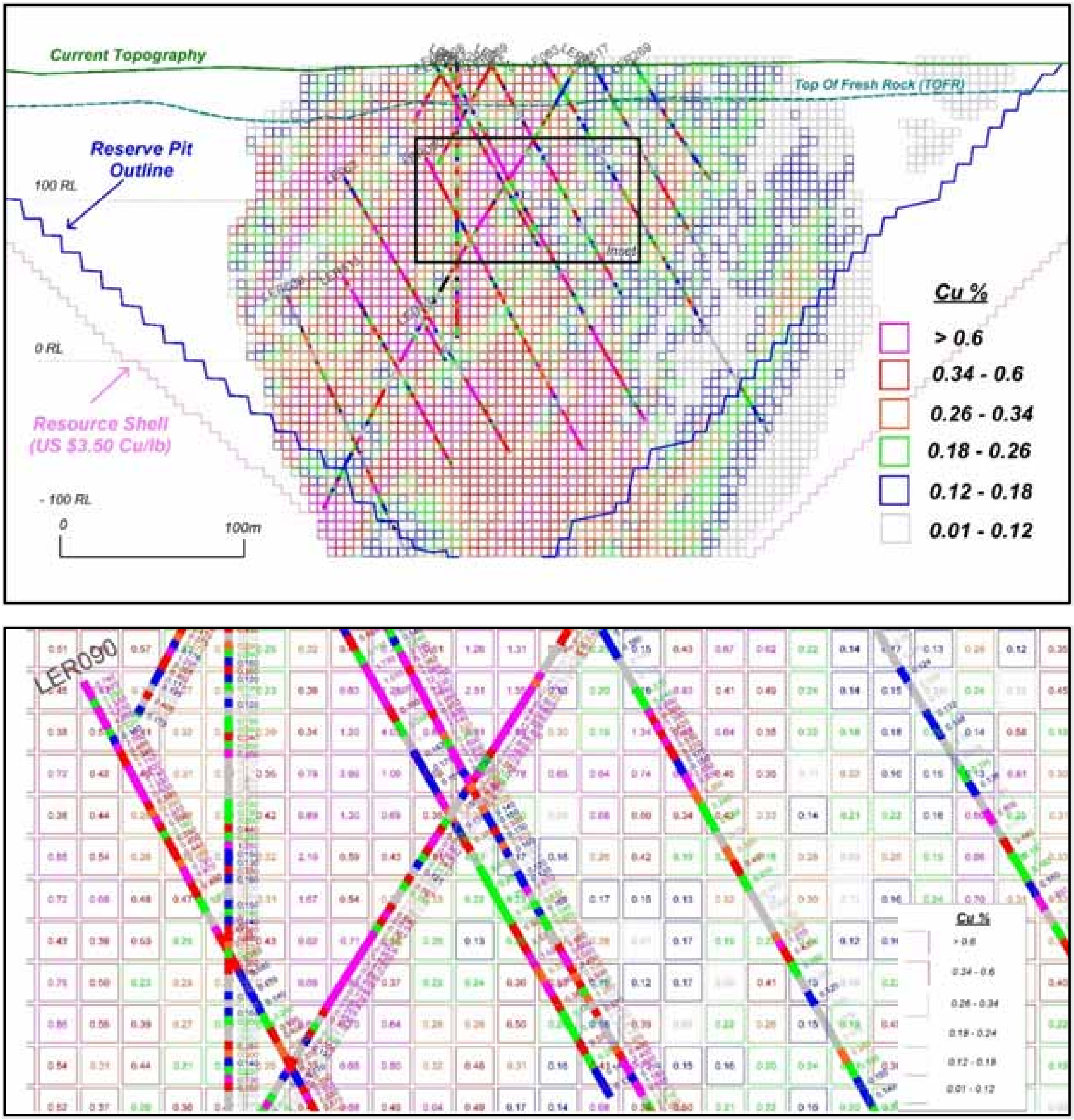


Figure 14-31: Cross-Section through the North End of the Eva Deposit with Block Grades and Drill Hole Composites from LED1006, Drilled Post-Estimation



Note: Illustrates Colour-Coded Drill Hole Composites Relative to block grades in upper image and a close-up (box) with printed grades in lower.

Figure 14-32: Cross-Section 7,772,000N in Little Eva Deposit

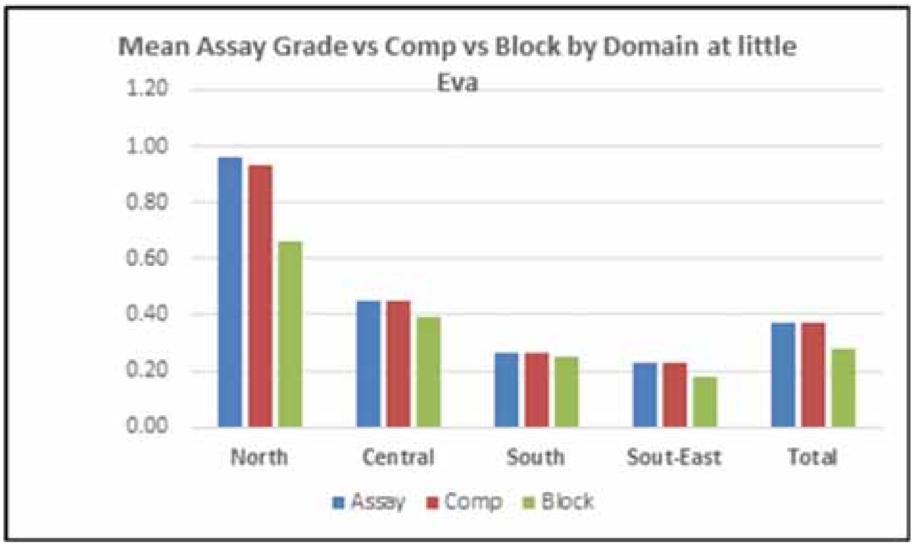


Figure 14-33: Mean Assay, Composite, and M&I Block Grades for the Different Resource Domains in the Little Eva Deposit

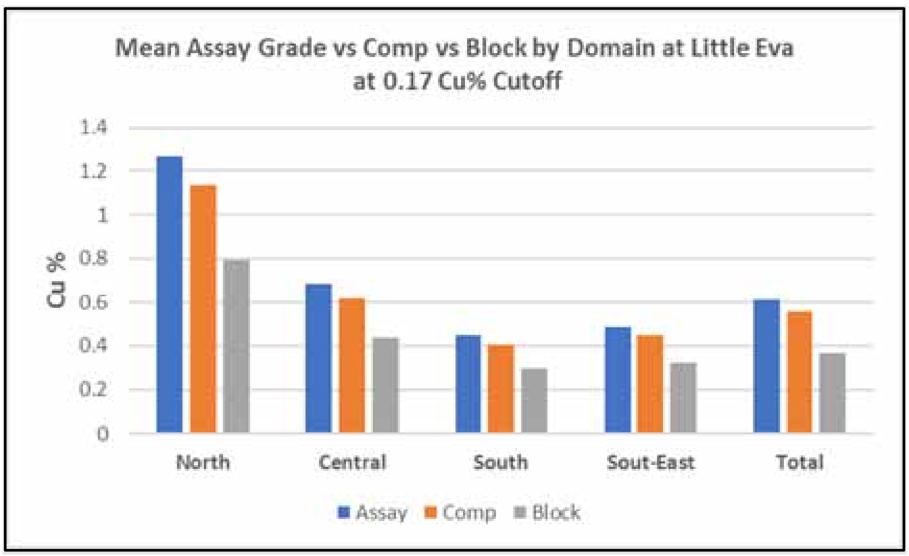


Figure 14-34: Mean Assay, Composite, and Block Grades for Domains in the Little Eva Deposit at a 0.17% Cu Cut-off Grade

Different interpolation methods were compared using grade-tonnage curves for the Little Eva deposit as shown in Figure 14-35. The points on the curves are tonnages that correspond to a range of cut-off grades. The NN method of interpolation is where the block grade is assigned a value based on the closest composite to the block. NN-type interpolations almost always produce the highest grades of the different interpolation methods and are generally only used for comparison purposes. The Optiro 2017 resource estimate was made using multiple indicator kriging (MIK) which allows for modelling of mineralization grade continuity at different grade ranges. In addition, the Optiro estimate used uniform conditioning, which is a method of ensuring that block grades have the same grade distribution as the composite grades, which in this estimate were based on a1 m composite length).

The OK and ID^2 interpolations produced similar 'grade-tonnage' curves (Figure 14-35), both of which have lower grades than the grade-tonnage values from estimates using the NN and MIK methods. The MIK with uniform conditioning method produces slightly higher grades than the NN method, which likely reflects the difference in composite lengths (1 m vs. 2.5 m). While the MIK method produces block grades that may match the distribution of 1m composite grades, CMMC believes that this incorporates a degree of selectivity that cannot be matched in a bulk mining situation, and that the OK method is a better indicator of what mining will achieve assuming normal mining practices. It is concluded that the OK interpolation in the Little Eva deposit yields a reliable estimate for mine planning and financial analysis.

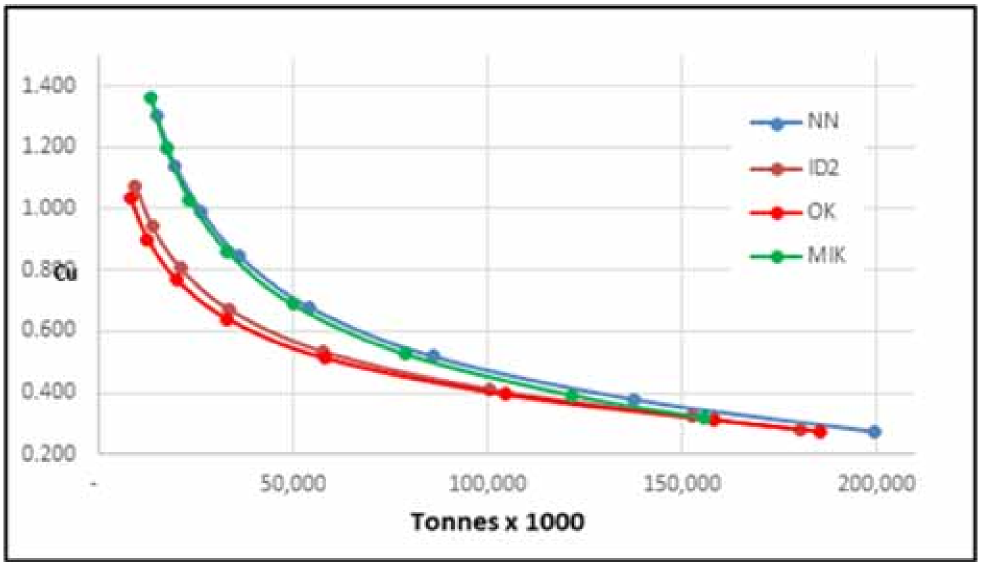


Figure 14-35: Tonnage vs. Cu Cut-off Grade Curves for the Little Eva Deposit Using four Estimation Methods

ID^2 interpolations produce similar results to OK methods, particularly when the same composite and block sizes are used; the variation in composite grades is reasonable (CoV <1.7), and the drill hole data is not excessively clustered. As these conditions were generally met by the other deposits, it is reasonable that resource estimates in this report were similar to previous methods where kriging and or other methodology had been used. Like the comparisons of resource estimates for the Little Eva deposit, the copper grades of the reported herein from the other Eva Copper Project deposits are slightly lower than previous estimates particularly those where uniform conditioning was applied.

Overall, the new resource estimates for all the deposits within the Eva Copper Project are similar to previous estimates, generally with slight reductions in grade, but with similar or slightly increased tonnages (changes in classification notwithstanding). The grade reduction is primarily a function of using larger composite size and interpolation methodology that incorporates anticipated mining dilution. Additionally, changes in estimation methodology has resulted in some movement between resource classifications.

15 MINERAL RESERVE ESTIMATES

The conversion of Mineral Resources to Mineral Reserves requires knowledge gathered through pit optimization, pit design, economics, and other modifying parameters.

The Mineral Reserves were calculated based on net smelter return (NSR) cut-off values for each pit area; Little Eva, Turkey Creek, Bedford, Lady Clayre, Ivy Ann, Blackard, and Scanlan.

In accordance with CIM classification guidelines, only Measured and Indicated Mineral Resource categories are converted to Proven and Probable Mineral Reserves, respectively, through inclusion within the open-pit mining limits. Inferred Mineral Resources are treated as waste, with an assigned grade of zero.

Mineral Reserves are most sensitive to the estimated grade, copper and gold prices, and the metallurgical recoveries for copper and gold. The Eva Copper Project has already received most of the required permits, with any additional amended permits expected in a timely manner. Copper Mountain Mining Corp. (CMMC) is currently composing an amendment to the current permit, because of the increased mill production and additional areas of disturbance.

A summary of Mineral Reserves is shown in Table 15-1. The Mineral Reserves were prepared by CMMC in the Fall of 2019, and have been independently audited by Mr. Stuart Collins, P.E. to reflect the Mineral Reserves as of January 31, 2020.

15.1 Summary

The Eva Copper Project has a Mineral Reserve of 171 Mt grading 0.46% Cu and 0.05 g/t Au for 1.72 Blb contained copper and 260,000 oz contained gold. Little Eva, Blackard, Scanlan, and Turkey Creek account for approximately 48%, 30%, 10%, and 7% of the copper in the Mineral Reserve, respectively.

All deposits have sulphide-only ore tonnages classified as either Proven or Probable Mineral Reserves, and additional Inferred Mineral Resources that are not included in the Mineral Reserves and LOM schedule. Oxide materials have not been included in the Mineral Reserves. All Mineral Reserves are classified and reported in accordance with the 2011 CIM Standards (CIM, 2011). CMMC is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Mineral Reserves estimate.

Mineral Reserves for Little Eva, Ivy Ann, Bedford, Lady Clayre, Blackard, and Scanlan were previously estimated by Optiro (an Altona consultant) in 2012. Turkey Creek Mineral Reserves were previously estimated by Altona in 2016, based on the mining inventory by Orelogy (an Altona consultant). All Mineral Resource models and Mineral Reserves were updated by CMMC in 2018 and 2019.

Rounding may result in apparent differences when summing tonnes, grades, and contained metal content. Tonnage and grade measurements are in metric units. Gold grades are reported in grams per tonne (g/t), and copper grades are reported in percentage total copper (% Cu).

Table 15-1: Eva Copper Project Mineral Reserves, January 31, 2020

Deposit	Mineral Reserve Classification	Cut-off Value (US$/t)	Ore Tonnes (kt)	Cu Grade (% Cu)	Au Grade (g/t)	Total Cu Pounds (Mlb)	Total Au Ounces (koz)
Little Eva	Proven	8.95	53,907	0.40	0.07	480	126
Lady Clayre	Proven	10.32	2,648	0.46	0.19	27	16
Ivy Ann	Proven	11.44	685	0.44	0.09	7	2
Bedford	Proven	9.35				-	-
Blackard	Proven	9.35	22,951	0.58		295	-
Scanlan	Proven	10.32	6,279	0.72		100	-
Turkey Creek	Proven	8.95	6,151	0.49		66	-
Total	**Proven**	**Varies**	**92,623**	**0.48**	**0.05**	**975**	**144**
Total for Gold Grade only	**Proven**		57,241	0.41	0.08	513	144
Little Eva	Probable	8.95	43,805	0.36	0.06	348	91
Lady Clayre	Probable	10.32	831	0.45	0.21	8	6
Ivy Ann	Probable	11.44	1,640	0.42	0.09	15	5
Bedford	Probable	9.35	2,863	0.56	0.15	35	14
Blackard	Probable	9.35	19,756	0.52		228	-
Scanlan	Probable	10.32	4,987	0.58		64	-
Turkey Creek	Probable	8.95	4,544	0.45		45	-
Total	**Probable**	**Varies**	**78,425**	**0.43**	**0.05**	**743**	**115**
Total for Gold Grade only	**Probable**		49,139	0.37	0.07	406	115
Little Eva	Proven + Probable	8.95	97,712	0.38	0.07	828	217
Lady Clayre	Proven + Probable	10.32	3,479	0.45	0.20	35	22
Ivy Ann	Proven + Probable	11.44	2,325	0.43	0.09	22	7
Bedford	Proven + Probable	9.35	2,863	0.56	0.15	35	14

Deposit	Mineral Reserve Classification	Cut-off Value (US$/t)	Ore Tonnes (kt)	Cu Grade (% Cu)	Au Grade (g/t)	Total Cu Pounds (Mlb)	Total Au Ounces (koz)
Blackard	Proven + Probable	9.35	42,707	0.56	0.00	523	-
Scanlan	Proven + Probable	10.32	11,266	0.66	0.00	164	-
Turkey Creek	Proven + Probable	8.95	10,695	0.47	0.00	112	-
Total	**Proven + Probable**	**Varies**	**171,047**	**0.46**	**0.05**	**1,718**	**260**
Total for Gold Grade Only	**Proven + Probable**		106,380	0.39	0.08	919	260

Notes: **1.** CIM Definition Standards were followed for Mineral Reserves. **2.** Mineral Reserves were generated using the January 31, 2020 mining surface. **3.** Mineral Reserves are reported at an NSR cut-off value of $8.95/t for Little Eva and Turkey Creek, $9.35/t for Bedford and Blackard, $10.32/t for Lady Clayre and Scanlan, and $11.44/t for Ivy Ann. **4.** Mineral Reserves are reported using long-term copper and gold prices of $2.75/lb and $1,250/oz, respectively. **5.** Average process recoveries used in pit optimization ranged from 90% to 93% for copper sulphide, 63% for native copper, and 78% for gold were used for all deposit areas. **6.** Little Eva, Turkey Creek, Bedford, and Lady Clayre have an equivalent 5.3% NSR royalty; Ivy Ann has an equivalent 5.8% royalty. **7.** Blackard, Scanlan, and Turkey Creek do not contain gold. **8.** Totals may show apparent differences due to rounding.

15.2 Mineral Reserve Development

Listed below are most of the major changes to the Project since 2017, including:

- New Mineral Resource estimates or geological/geotechnical models for the Little Eva, Turkey Creek, Bedford, Lady Clayre, Blackard, Scanlan, and Ivy Ann deposit areas
- Two re-estimations of initial capital costs by Merit
- An increase in engineering and construction costs, and changes to macro-economic assumptions
- Self-mining by CMMC for the Eva deposits (Little Eva, Blackard, Scanlan, Turkey Creek, Bedford, and Lady Clayre).

15.2.1 Little Eva Deposit

CMMC optimized and designed the Little Eva pit as a part of the 2019 Feasibility Study, which is the basis of the Little Eva Mineral Reserve.

The final pit is approximately 1,700 m long and 950 m wide, and the final depth is approximately 310 m. The initial pre-strip of the Little Eva pit is approximately 10.5 Mt prior to accessing the first ore between 160.0 mASL and 142.5 mASL. The overall average strip ratio for the Little Eva pit is approximately 1.55 tonnes waste to one tonne ore (w:o) excluding pre-strip, and approximately 1.8 w:o including pre-strip.

The mining of the Little Eva deposit will be carried out in six stages with a starter pit targeting the higher-grade North and Central domains, a pushback after two to three years, and a final pit targeting the southeastern domain.

Safety and creek diversion bunds will be required around the pit perimeter. Some of these will be multipurpose, being used for creek diversion, safety, and pit reclamation. Figure 15-1 is a combination ultimate pit plan and representative cross-section of the Little Eva pit.

15.2.2 Turkey Creek Deposit

The Turkey Creek pit was optimized and designed by Orelogy in 2016 using similar parameters to those used for Little Eva. A similar approach was taken by CMMC in 2018, using Little Eva pit parameters for the Turkey Creek deposit.

The Turkey Creek pit design has a Stage 1 starter pit to access the ore quickly, with Stage 2 as a single cutback to the full depth of the optimized pit. Ore is to be hauled just under a kilometre around the plant to access the run-of-mine (ROM) pad. The Stage 1 pit has a depth of around 110 m, and was constrained to allow sufficient mining width for the subsequent cutback. The last stage of mining takes the pit base to 170 m deep. The final Turkey Creek pit is approximately 1,100 m long and 400 m wide. The average strip ratio is 2.2:1 w:o. Figure 15-2 is a combination ultimate pit plan and representative cross-section of the Turkey Creek pit.

15.2.3 Bedford Deposit

Bedford is composed of the North and South pits, 800 m apart. By using spiral ramps, ore can be accessed with the least amount of waste stripping. Bedford North is 630 m long and 260 m wide, and the final depth is 110 m. Bedford South is 340 m long and 200 m wide, and the final depth is 80 m.

The average strip ratio (North + South) is 4.4:1 w:o. Figure 15-3 is a combination ultimate pit plan and representative cross-section of the Bedford North pit.

The Bedford Mineral Resource was updated in 2018, and the pit designs were completed.

15.2.4 Lady Clayre Deposit

A new geological model has been developed for Lady Clayre, and resource estimation within the new model was completed by CMMC personnel. The Lady Clayre pit design comprises three small pits that merge near surface. Located approximately 20 km south of the Eva Copper Project processing plant, the pits have been designed in such a way that they can be mined separately. The combined Lady Clayre pit is 730 m long, 350 m wide, and 100 m deep. The average strip ratio is 4.3:1 w:o. Figure 15-4 is a combination ultimate pit plan and representative cross-section of the Lady Clayre pits.

15.2.5 Ivy Ann

Ivy Ann is a single-stage pit located approximately 36 km from the processing plant. The pit will be mined using a spiral ramp. Ivy Ann is 490 m long and 350 m wide, and the final depth is 120 m. The average strip ratio is 3.4:1 w:o. Figure 15-5 is a combination ultimate pit plan and representative cross-section of the Ivy Ann pit.

15.2.6 Blackard Deposit

A new geological model has been developed for Blackard, and resource estimation within the new model was completed by CMMC personnel. The Blackard pit design is comprised of one large pit with two pit bottoms. Located approximately 6.8 km south of the Eva Copper Project processing plant, the pit is composed of copper ore only; no extractable gold values exist at this pit. The combined Blackard pit is 1,900 m long, 657 m wide, and 235 m deep. The average strip ratio is 3.1:1 w:o. Figure 15-6is a combination ultimate pit plan and representative cross-section of the Blackard pit.

15.2.7 Scanlan Deposit

Scanlan is composed of the North and South pits, 550 m apart. By using spiral ramps, ore can be accessed with the least amount of waste stripping. Scanlan North is 340 m long and 340 m wide, and the final depth is 80 m. Scanlan South is 740 m long and 465 m wide, and the final depth is 165 m. The average strip ratio (north + south) is 2.7:1 (waste to ore). Figure 15-7 is a combination ultimate pit plan and representative cross-section of the Scanlan North pit.

Located approximately 18.8 km south of the Eva Copper Project processing plant and north of the Lady Clayre deposit, the pit is composed of copper ore only; no extractable gold values exist at this pit. The Scanlan Mineral Resource was updated in 2019, and the pit designs were completed.

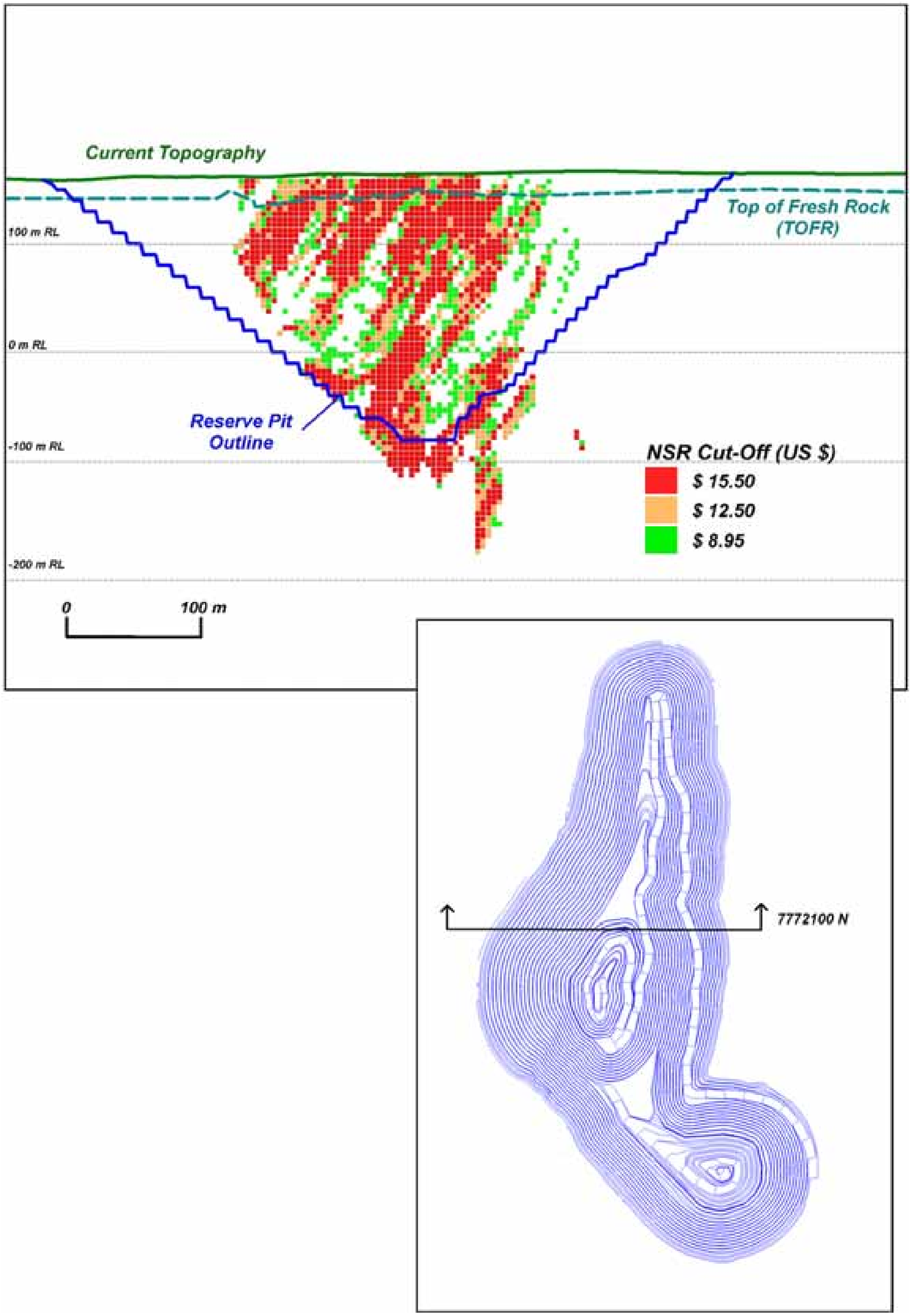


Figure 15-1: Little Eva Ultimate Pit Plan and Cross-Section

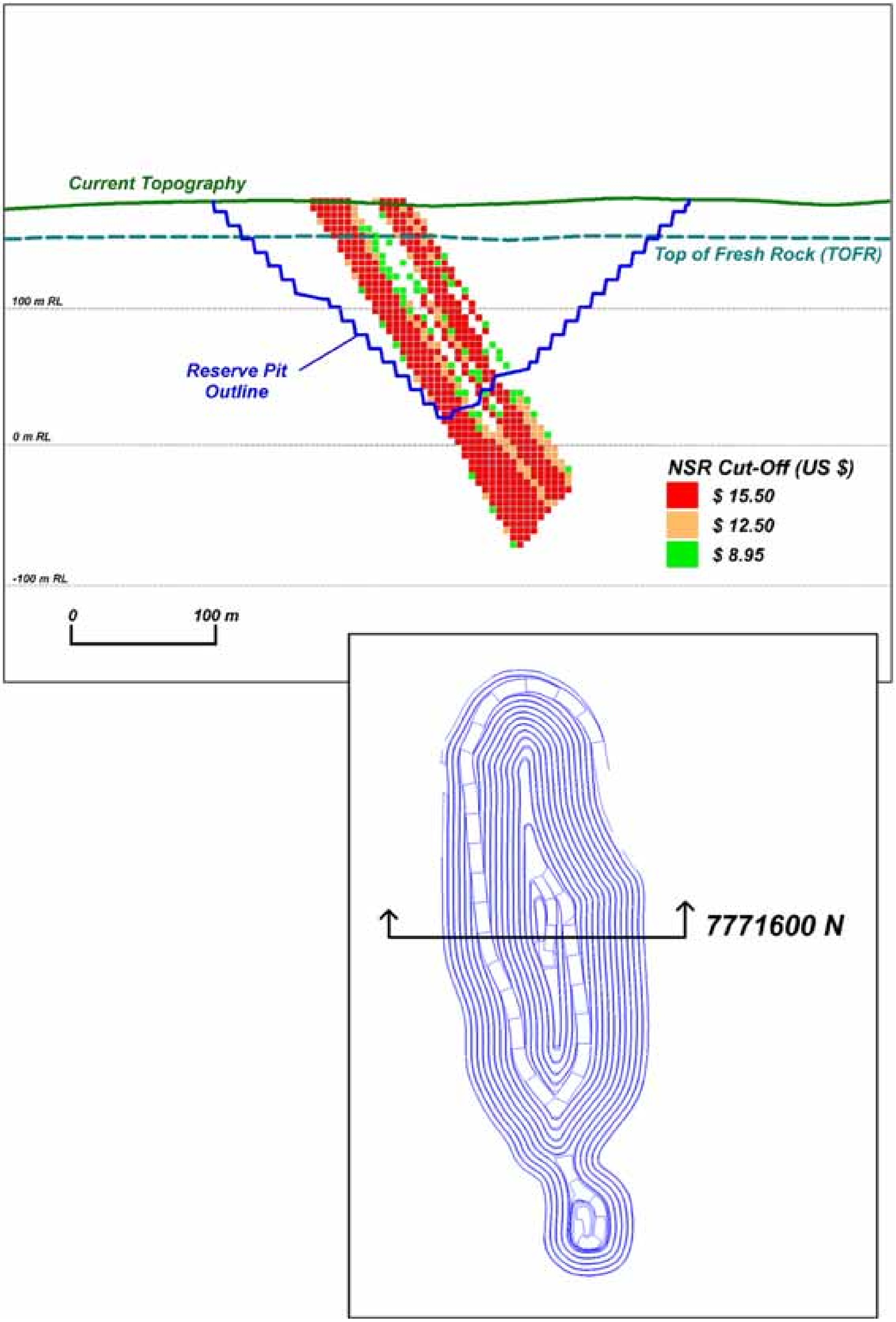


Figure 15-2: Turkey Creek Ultimate Pit Plan and Cross-Section

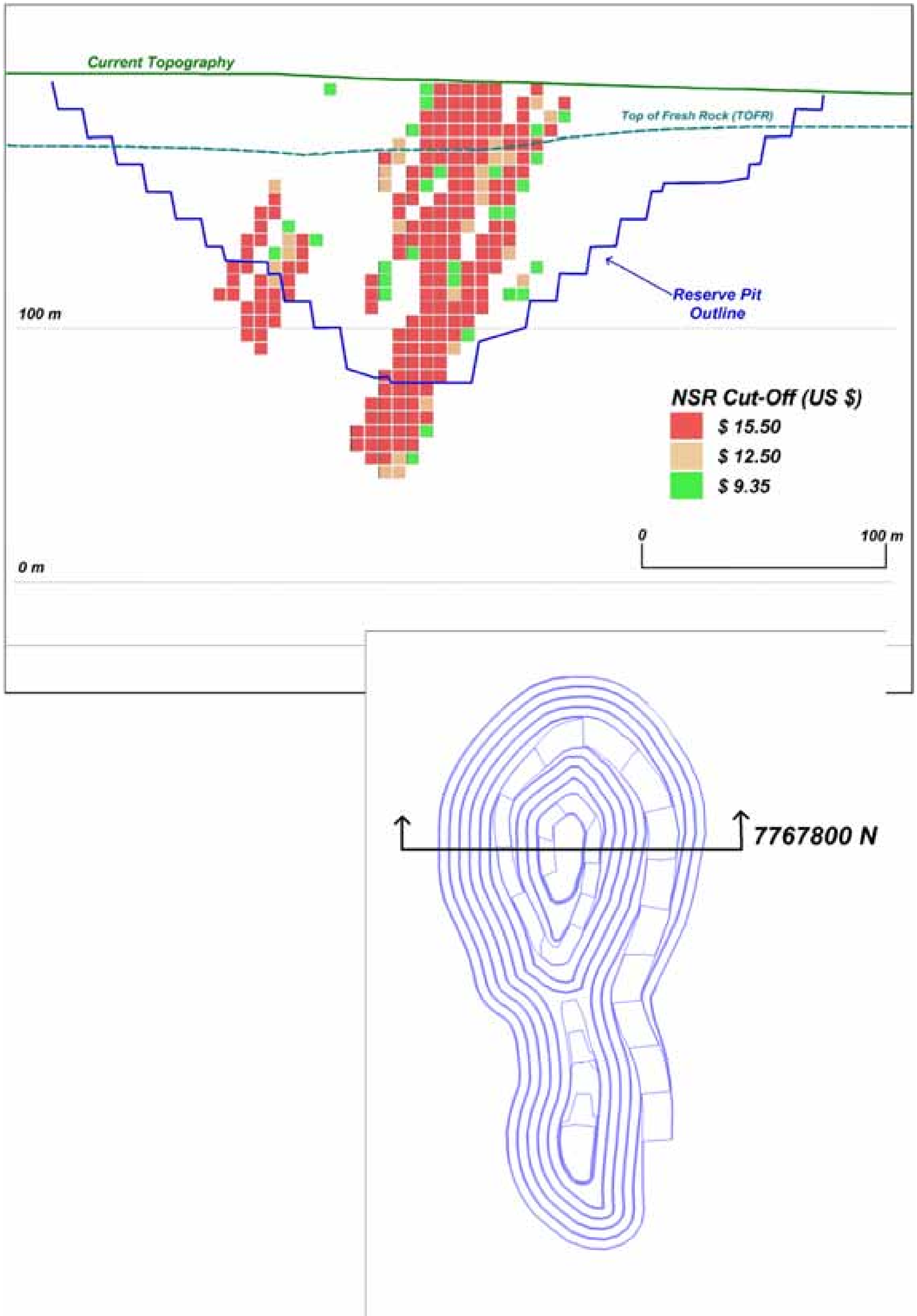


Figure 15-3: Bedford North Ultimate Pit Plan and Cross-Section

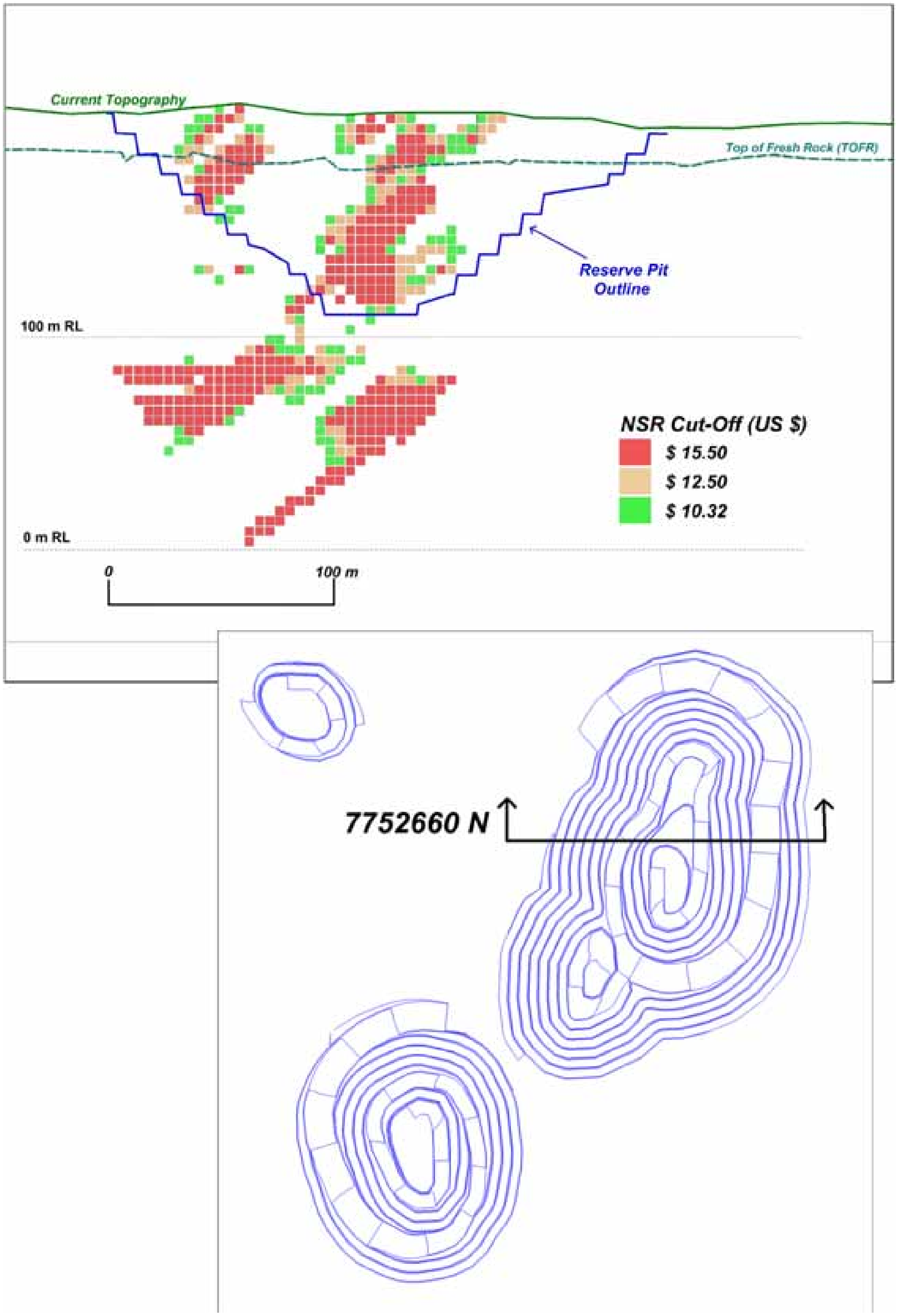


Figure 15-4: Lady Clayre Ultimate Pit Plan and Cross-Section

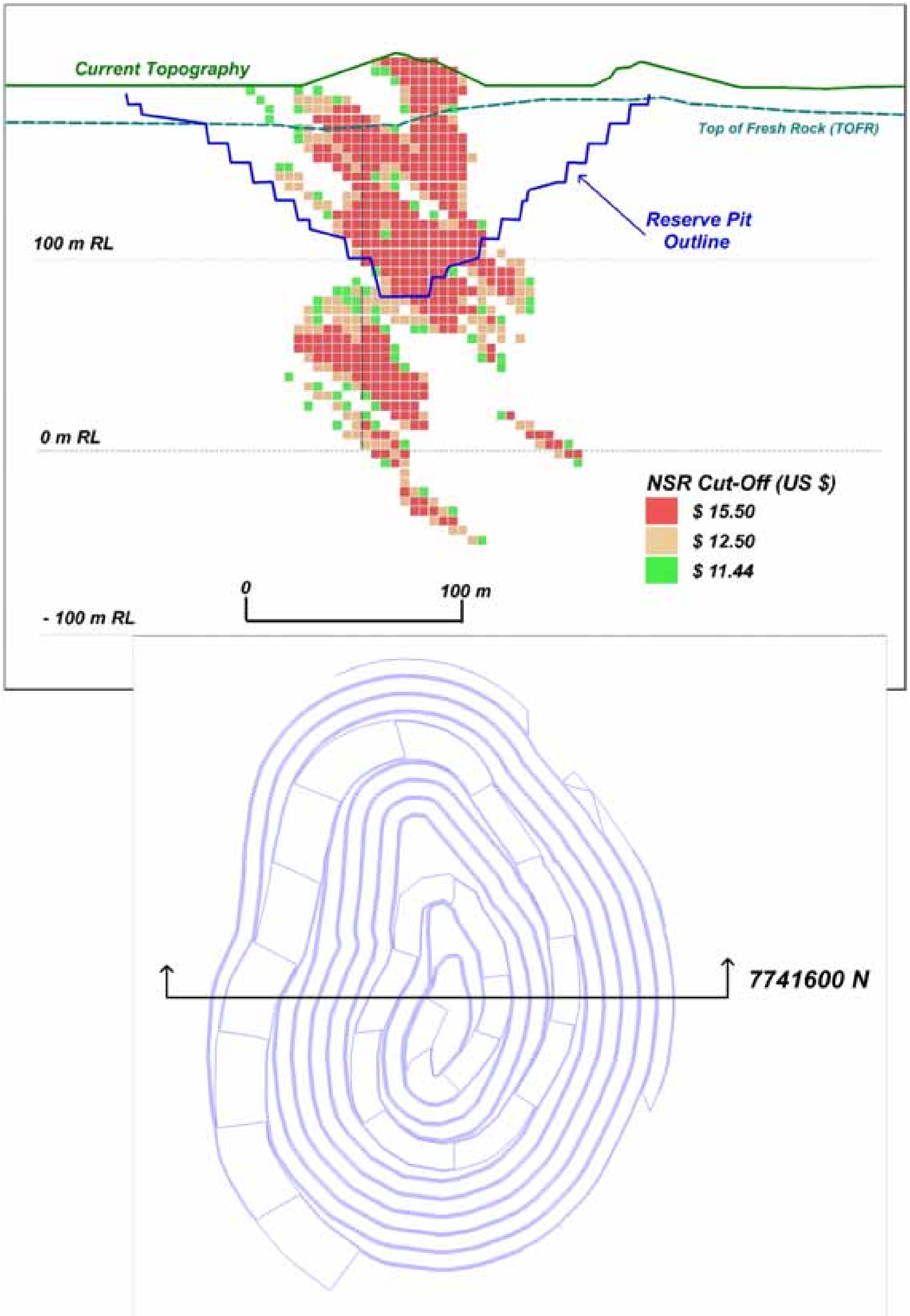


Figure 15-5: Ivy Ann Ultimate Pit Plan and Cross-Section

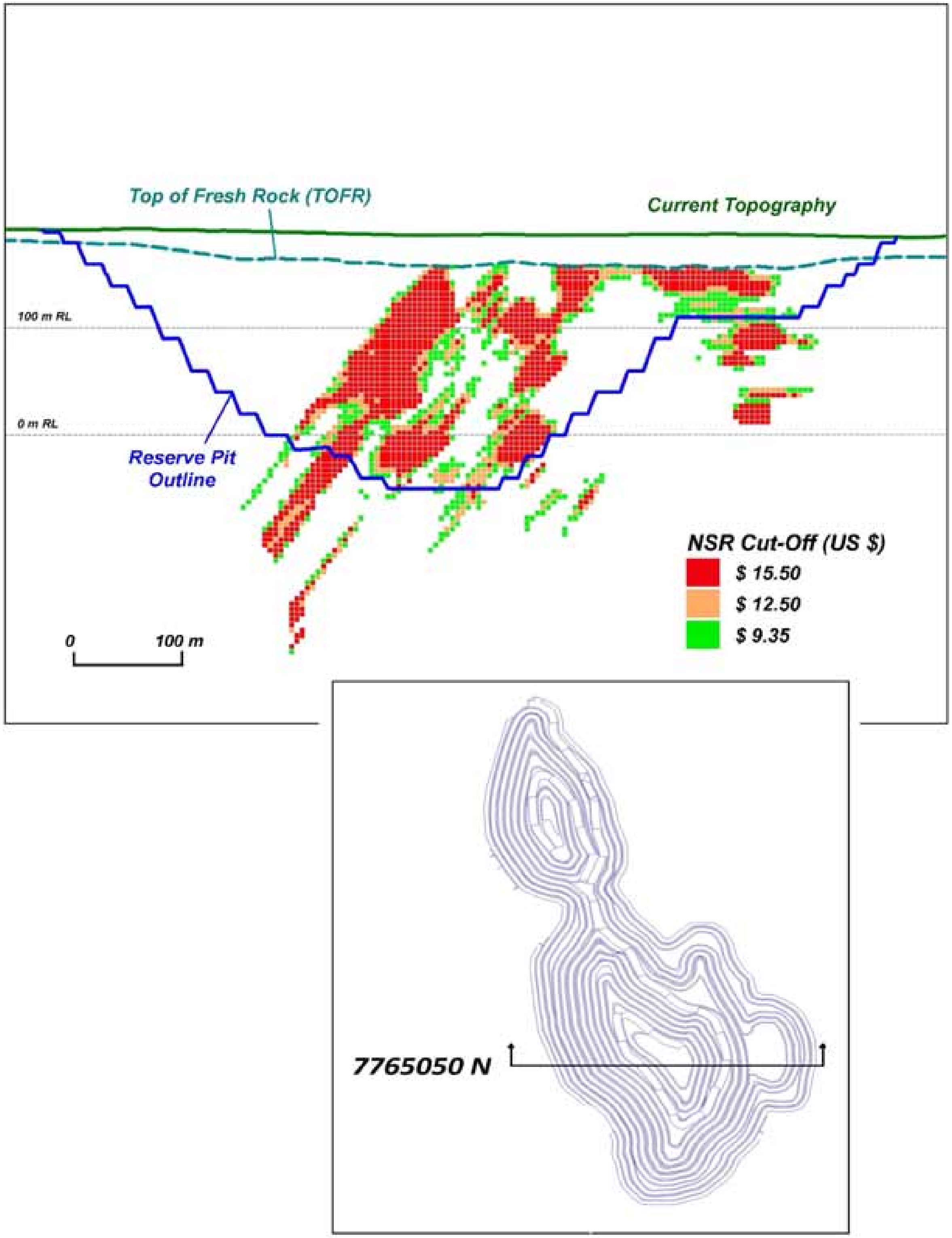


Figure 15-6: Blackard Ultimate Pit Plan and Cross-Section

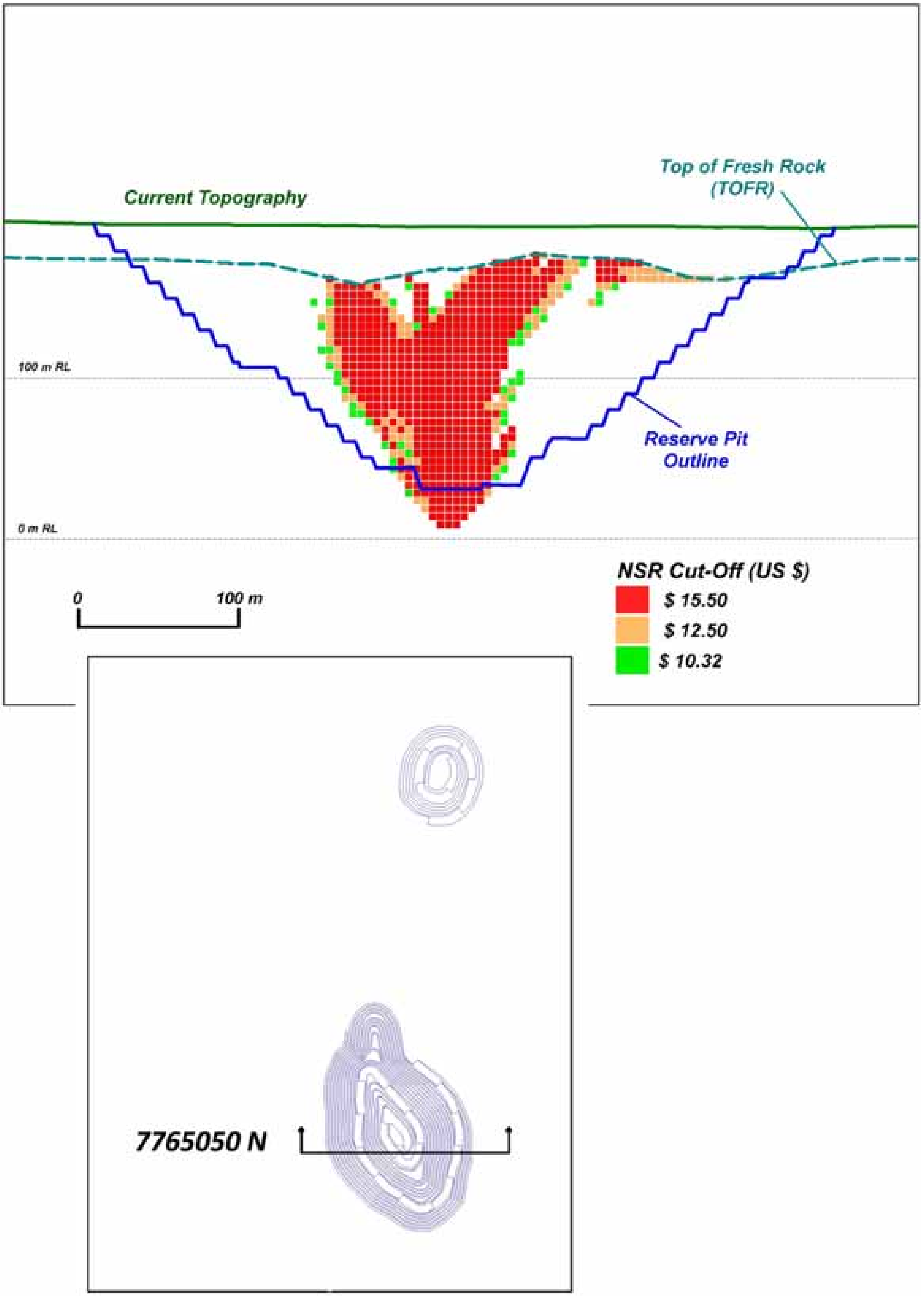


Figure 15-7: Scanlan Ultimate Pit Plan and Cross-Section

15.3 Density

The average material bulk densities used for the Mineral Reserves are listed in Table 15-2.

Table 15-2: Little Eva Bulk Densities by Deposit Area, January 31, 2020

Input Factor	Bulk Densities (t/m^3)
Overburden	1.50
Oxide	2.50
Intrusives, Metasediments, and Volcanics	2.63–2.80

15.4 Dilution and Mining Recovery

CMMC considers that the compositing and modelling processes used incorporate adequate dilution. The mine will be a moderately-sized operation, in which most of the ore-grade material is hauled to a crusher for crushing, or to a low-grade stockpile for future processing at the end of the Project's life.

The resource estimation process attempts to estimate the mineable tonnage and grade based on the dimensions of a selective mining unit (SMU), which is regarded as representative of what is practically achievable during actual mining. The SMU is the smallest volume that can be classified as either ore or waste. If the average grade of the SMU is greater than the cut-off value, the SMU is classified as ore.

For the Mineral Resource estimate, the chosen SME (or block size) of 5 m by 5 m by 5 m (X, Y, Z) can be considered diluted as they are estimated using all data within a broad mineralized envelope defined by an approximate 0.0% Cu cut-off grade, which is based on geological and mineralization limits.

The estimation process incorporates a change of support to the large diluted panels to predict the likely grade-tonnage distribution at SMU selectivity. An advantage of this method is that mining dilution is implicit within the predicted tonnage and grade distribution of the SMUs, as the distribution of grades is conditioned by the diluted panel grades. The SMU dimensions are large enough to incorporate sufficient dilution for the geometry of the Eva Copper Project mineralization. The implicit inclusion of dilution in the SMU will also result in some ore loss from the resource, where the grade of the SMU drops below the economic cut-off value. Additionally, a further "information effect" modification was made to the change of support to allow for imperfect grade estimation and mining selection, because of an assumed grade control spacing of 5 m by 5 m by 5 m.

Because of the adopted resource estimation methodology, no further modifying factors for dilution or ore loss were applied to the resource block model for use in mine planning and reporting.

15.5 Cut-off Net Smelter Return Value

NSR values were used for the reporting of Mineral Reserves. This method was used because there are two metals produced from the Eva Copper Project, copper and gold. Metal prices used for Mineral Reserves are based on consensus and long-term forecasts from banks, financial institutions, and other sources. For Mineral Resources, metal prices used are slightly higher than those for Mineral

Reserves. The NSR cut-off values used as part of the estimation of Eva Copper Project Mineral Reserves and Mineral Resources are as follows:

- Open Pit Mineral Reserve, reported at NSR cut-off values by pit(s) of: $8.95/t for Little Eva and Turkey Creek; $9.35/t for Bedford and Blackard; $10.32/t for Lady Clayre and Scanlan, and $11.44/t for Ivy Ann. Cut-off costs vary due to the one-way ore haul distances of approximately 6.90 km for Bedford, 6.80 km for Blackard, 18.80 km for Scanlan, 19.65 km for Lady Clayre, and 35.85 km for Ivy.
- Open Pit Mineral Resources (sulphide only), reported at a copper cut-off grade of 0.17% Cu.

15.6 Pit Optimization

Economic constraints utilize a minimum NSR cut-off value, which is compared against each block (SMU) within the model. If the block grade is above the designated cut-off NSR value, then the net block value before tax is calculated for the block using the parameters and appropriate block tonnages estimated. If the block grade is below cut-off value, the block is designated as waste and assigned a cost of mining for the block (5 m by 5 m by 5 m) of material. Only Measured and Indicated Mineral Resources were used in the pit optimization. In addition, all oxide material was treated as waste.

There are two cut-offs typically used in the mining industry: breakeven and internal. The Project Whittle pit shells (Little Eva, Turkey Creek, Bedford, Lady Clayre, Ivy Ann, Blackard, and Scanlan pits) were generated using a break-even cut-off, which indicates that the mining cost was part of the cut-off calculation applied to every tonne of material mined. This method produces a more conservative economic shell as a design guide. After the break-even pit is designed with berms, batters, and roads, an internal cut-off is applied to the tonnage inside the designed pit. An internal cut-off removes the mining cost from the calculation, which slightly reduces the NSR cut-off value, and, in turn, increases the process tonnes. This method recognizes that within the breakeven pit, all material (ore and waste) must be mined. All mining scenarios were evaluated on a cash flow basis, both individually and collectively.

Whittle optimization is an iterative process and was performed using costs developed during previous Eva Copper Project studies, which were updated during CMMC's Feasibility Study in 2018. The ore will ultimately be processed for copper and gold, and the Whittle Lerchs-Grossmann (LG) pit shells for the Feasibility Study were generated using copper (at $2.75/lb) and gold (at $1,250/oz) as the revenue sources. The Whittle pit optimizations for the Eva Copper Project Feasibility Study used Measured and Indicated Resources only when developing the designed pits.

Open pit mining of each Mineral Resource area was analyzed using Whittle 4.7.2 pit optimization software, which utilized relevant design, process, cost, and revenue input parameters. The output from the software provides a series of progressively larger pit shells, each reporting grade, metal content, and NPV resulting from the increased tonnages mined. Comparison of the results for each pit shell assists with the determination of the optimum size and tonnage for the pit and provides the basis for preliminary pit design. The optimization process determines the parts of the orebody that can be mined economically, and the process can indicate the optimum location for a starter pit at the commencement of mining. A summary of the pit optimization parameters used for each deposit area is shown in Table 15-3.

Table 15-4 through Table 15-10are summaries of the Whittle LG pit optimization analyses at variable metal prices or revenue factors, which are further used for pit designs. The base case copper and gold prices of $2.75/lb and $1,250/oz were used to establish the final pits, and NSR cut-off values for the five pit areas are listed below. Process costs (as an incremental cost) for the satellite pits increase due to the estimated increased haulage distance to the processing plant. The ore haul from Bedford is approximately 6.90 km, from Blackard approximately 6.80 km, from Scanlan approximately 18.80 km, from Lady Clayre approximately 19.65 km, and from Ivy Ann approximately 35.85 km. The resulting NSR cut-off values by pit are as follows:

- Little Eva and Turkey Creek..$8.95/t
- Bedford (North and South) and Blackard..$9.35/t
- Lady Clayre (West, North, South) and Scanlan....................................$10.32/t
- Ivy Ann $11.44/t

After the ultimate Mineral Reserve shells were determined, the Whittle data were used to help guide and develop the ultimate pit designs, which include ramp and bench designs. Phasing for the Little Eva, Turkey Creek, Bedford, Lady Clayre, and Ivy Ann pits was then based on the Whittle analyses, ultimate pit designs, and individual deposit economics.

Table 15-3: Eva Pit Optimization Inputs, August 31, 2018

Input Factor	Unit	Little Eva	Turkey Creek	Bedford	Lady Clayre	Ivy Ann	Blackard	Scanlan
Block Size (I,j,k)	m	(5,5,5)	(5,5,5)	(5,5,5)	(5,5,5)	(5,5,5)	(5,5,5)	(5,5,5)
Model Origin (lower left)								
Easting	m	410,100	412,000	414,721	409,132.15	425,100	411,800	411,900
Northing	m	7,771,000	7,770,750	7,765,598	7,751,523.37	7,741,000	7,764,300	7,753,650
Elevation Bottom	m	-400	-350	0	-600	-100	-500	-240
Model Rotation	deg.	0	0	0	0	0	0	0
Number of Blocks (x,y,z)								
x direction (columns)	num.	280	260	100	272	380	320	158
y direction (rows)	num.	380	400	579	352	720	500	380
z direction (levels)	num.	116	120	42	200	76	180	100
Model Variables Used								
		Rock Type, Density, Cu, Au, Class, NSR2	Rock Type, Density, Cu, Class, NSR2	Rock Type, Density, Cu, Au, Class, NSR2	Rock Type, Density, Cu, Au, Class, NSR2	Rock Type, Density, Cu, Au, Class, NSR2	Rock Type, Density, Cu, Class, NSR2	Rock Type, Density, Cu, Class, NSR2
Mineral Classes used in Optimization	M/I/I	M&I only	M&I only	M&I only	M&I only	M&I only	M&I only	M&I only
Incremental Bench Mining Cost Calculation	Formula	e.g., Little Eva: R(IZ,-0.0035*IZ+1.2786,116,1) Turkey Creek similar, N/A to others						
Geotechnical Parameters								
Hanging Wall Zone	deg.	52	43	43	43	43	43	43
Footwall Zone	deg.	43 – 47	43	43	43	43	43	43
Target Copper Concentrate Grade	%Cu	25	25	25	25	25	25	25
Copper Price	$/lb	2.75	2.75	2.75	2.75	2.75	2.75	2.75
Gold Price	$/oz	1,250.00	N/A	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00

Input Factor	Unit	Little Eva	Turkey Creek	Bedford	Lady Clayre	Ivy Ann	Blackard	Scanlan
Copper Recoveries	%	93	93	93	93	93	93, 63	93, 63
Gold Recovery	%	78	N/A	78	78	78	78	78
Payable Copper	%	96	96	96	96	96	96	96
Payable Gold (1 g/t<Au<3 g/t)	%	90	N/A	90	90	90	90	90
Payable Gold (3 g/t<Au<5 g/t)	%	92	N/A	92	92	92	92	92
Payable Gold (5 g/t<Au<7 g/t)	%	94	N/A	94	94	94	94	94
Copper Concentrate Grade	%	25	25	25	25	25	25	25
Processing Rate	t/d	26,000	26,000	26,000	26,000	26,000	26,000	26,000
Moisture	%	9	9	9	9	9	9	9
Concentrate Loss	%/wmt	0.15	0.15	0.15	0.15	0.15	0.15	0.15
Copper Refining Charge	$/lb	0.08	0.08	0.08	0.08	0.08	0.08	0.08
Gold Refining Charge	$/oz	5.00	N/A	5.00	5.00	5.00	5.00	5.00
Costs								
Mining Cost (Average LOM)	$/t – mine	2.01	2.01	2.01	2.01	2.01	2.01	2.01
Incremental Mine Cost	$/t/10 m bench	0.0035	0.0035	0.0035	0.0035	0.0035	0.0035	0.0035
Milling Cost	$/t – mill	7.45	7.45	7.85	8.82	9.94	7.85	8.82
G&A Cost	$/t – mill	1.50	1.50	1.50	1.50	1.50	1.50	1.50
Milling + G&A Cost	$/t – mill	8.95	8.95	9.35	10.32	11.44	9.35	10.32
Exchange Rate (nominal)	AU$:US$	1.32	1.32	1.32	1.32	1.32	1.32	1.32
Treatment Cost	$/dmt	80.00	80.00	80.00	80.00	80.00	80.00	80.00
Total Offsite Cost	**$/wmt concentrate**	**113.00**	**113.00**	**113.00**	**113.00**	**113.00**	**113.00**	**113.00**
Insurance	%	0.175	0.175	0.175	0.175	0.175	0.175	0.175
Equivalent Royalties								
Royalty	%	5.3	5.3	5.3	5.3	5.8	5.3	5.3

Table 15-4: Little Eva Pit Optimization Results, August 31, 2018

Pit Shell	NSR Cut-off ($/t)	Total Tonnes (kt)	Waste Tonnes (kt)	Ore Tonnes (kt)	Strip Ratio (w:o)	Cu Grade (%Cu)	Au Grade (g/t)	NSR2 Value ($/t)	Cu Tonnes (kt)	Au Ounces (koz)
1	2.69	608	237	144	1.6	1.46	0.17	69.05	2.1	0.8
6	3.58	9,894	758	3,064	0.2	0.86	0.12	41.36	26.5	12.1
11	4.48	28,902	15,724	9,242	1.7	0.75	0.10	36.06	69.7	31.2
16	5.37	49,241	19,825	16,486	1.2	0.65	0.10	31.31	107.4	51.0
21	6.27	90,016	42,045	34,456	1.2	0.53	0.09	25.74	182.3	98.7
26	7.16	118,191	59,039	48,114	1.2	0.48	0.08	23.31	229.8	128.2
31	8.06	156,084	72,433	65,498	1.1	0.43	0.08	21.22	283.9	163.4
36	8.95	219,638	92,370	92,892	1.0	0.39	0.07	19.13	363.2	208.6
41	9.85	254,253	119,872	111,422	1.1	0.37	0.07	17.98	408.2	240.4
46	10.74	298,065	125,919	129,598	1.0	0.35	0.07	17.12	450.7	273.1
51	11.64	312,971	137,395	139,138	1.0	0.34	0.06	16.60	469.2	285.4
56	12.53	325,605	140,327	147,075	1.0	0.33	0.06	16.19	483.4	294.8
61	13.43	335,746	144,131	153,268	0.9	0.32	0.06	15.88	493.8	302.0
66	14.32	341,492	164,828	158,083	1.0	0.32	0.06	15.61	500.8	307.1
71	15.22	349,162	167,829	162,470	1.0	0.31	0.06	15.39	507.4	311.8
76	16.11	354,136	172,651	165,989	1.0	0.31	0.06	15.21	512.1	315.3
81	17.01	357,575	175,270	168,747	1.0	0.31	0.06	15.06	515.4	317.9
86	17.90	362,506	180,030	171,564	1.0	0.30	0.06	14.92	518.9	320.8

Notes: **1.** Base Case Whittle shell prices used for design were 2.75/lb Cu and $1,250/oz Au. **2.** NSR cut-off value of $8.95/t. **3.** Pit optimization used only Measured and Indicated Mineral Resources. **4.** Numbers may vary due to rounding.

Table 15-5: Turkey Creek Pit Optimization Results, August 31, 2018

Pit Shell	NSR Cut-off ($/t)	Total Tonnes (kt)	Waste Tonnes (kt)	Ore Tonnes (kt)	Strip Ratio (w:o)	Cu Grade (%Cu)	Au Grade (g/t)	NSR2 Value ($/t)	Cu Tonnes (kt)	Au Ounces (koz)
1	5.19	3,979	2,956	1,023	2.9	0.63	-	27.85	6.5	-
4	5.73	8,697	6,281	2,416	2.6	0.59	-	26.13	14.3	-
9	6.62	17,064	12,205	4,858	2.5	0.55	-	24.02	26.5	-
14	7.52	24,686	17,756	6,930	2.6	0.52	-	22.79	35.8	-
19	8.41	30,402	21,954	8,448	2.6	0.50	-	21.88	42.0	-
22	8.95	41,131	30,352	10,779	2.8	0.48	-	21.10	51.6	-
29	10.20	51,000	38,171	12,830	3.0	0.46	-	20.38	59.3	-
34	11.10	54,154	40,501	13,653	3.0	0.45	-	19.97	61.9	-
38	11.81	56,580	42,410	14,170	3.0	0.45	-	19.74	63.5	-
43	12.71	64,178	48,779	15,399	3.2	0.44	-	19.34	67.6	-
48	13.60	66,831	50,859	15,972	3.2	0.43	-	19.08	69.1	-
53	14.50	68,143	51,832	16,311	3.2	0.43	-	18.89	69.9	-
58	15.39	72,609	55,723	16,886	3.3	0.42	-	18.73	71.8	-
62	16.11	74,297	57,070	17,228	3.3	0.42	-	18.56	72.6	-
66	16.83	75,525	58,103	17,422	3.3	0.42	-	18.48	73.0	-
72	17.90	76,480	58,837	17,642	3.3	0.42	-	18.35	73.5	-

Notes: **1.** Base Case Whittle shell prices used for design were $2.75/lb Cu, and $1,250/oz Au. **2.** NSR cut-off value of $8.95/t. **3.** Pit optimization used only Measured and Indicated Mineral Resources. **4.** Numbers may vary due to rounding.

Table 15-6: Bedford Pit Optimization Results, August 31, 2018

Pit Shell	NSR Cut-off ($/t)	Total Tonnes (kt)	Waste Tonnes (kt)	Ore Tonnes (kt)	Strip Ratio (w:o)	Cu Grade (%Cu)	Au Grade (g/t)	NSR2 Value ($/t)	Cu Tonnes (kt)	Au Ounces (koz)
1	2.81	249	216	33	6.5	2.29	0.79	118.21	0.8	0.9
6	3.74	899	759	140	5.4	1.44	0.45	72.80	2.0	2.0
11	4.68	2,644	2,203	441	5.0	1.09	0.31	54.18	4.8	4.4
16	5.61	4,288	3,444	844	4.1	0.88	0.24	43.73	7.5	6.5
21	6.55	5,030	3,944	1,086	3.6	0.79	0.22	39.28	8.6	7.6
26	7.48	7,610	5,977	1,633	3.7	0.70	0.19	34.54	11.4	10.0
31	8.42	9,046	7,038	2,008	3.5	0.64	0.18	31.86	12.9	11.5
36	9.35	9,952	7,678	2,274	3.4	0.61	0.17	30.14	13.8	12.3
41	10.29	12,326	9,554	2,773	3.4	0.57	0.16	28.00	15.7	13.9
46	11.22	13,563	10,512	3,052	3.4	0.54	0.15	26.89	16.6	14.7
51	12.16	14,507	11,190	3,317	3.4	0.52	0.14	25.80	17.3	15.3
56	13.09	15,560	11,964	3,596	3.3	0.50	0.14	24.77	18.0	15.8
61	14.03	16,005	12,208	3,797	3.2	0.48	0.13	23.96	18.4	16.2
66	14.96	16,341	12,388	3,953	3.1	0.47	0.13	23.37	18.7	16.5
71	15.90	16,873	12,747	4,125	3.1	0.46	0.13	22.79	19.0	16.7
76	16.83	17,853	13,472	4,381	3.1	0.44	0.12	22.03	19.5	17.2
81	17.77	18,648	14,034	4,614	3.0	0.43	0.12	21.36	19.9	17.6
86	18.70	19,139	14,371	4,768	3.0	0.42	0.12	20.93	20.1	17.8

Notes: **1.** Base Case Whittle shell prices used for design were 2.75/lb Cu, and $1,250/oz Au. **2.** NSR cut-off value of $9.35/t. **3.** Pit optimization used only Measured and Indicated Mineral Resources. **4.** Numbers may vary due to rounding.

Table 15-7: Lady Clayre Pit Optimization Results, August 31, 2018

Pit Shell	NSR Cut-off ($/t)	Total Tonnes (kt)	Waste Tonnes (kt)	Ore Tonnes (kt)	Strip Ratio (w:o)	Cu Grade (%Cu)	Au Grade (g/t)	NSR2 Value ($/t)	Cu Tonnes (kt)	Au Ounces (koz)
1	5.37	508	419	89	4.7	0.81	0.34	43.49	0.7	1.0
5	6.19	646	522	125	4.2	0.76	0.30	40.27	1.0	1.2
10	7.22	6,250	5,150	1,101	4.7	0.60	0.30	33.63	6.6	10.8
15	8.26	8,877	7,096	1,781	4.0	0.55	0.25	30.23	9.9	14.3
20	9.29	10,862	8,481	2,381	3.6	0.51	0.22	27.69	12.2	16.9
25	10.32	14,207	11,061	3,146	3.5	0.48	0.20	25.78	15.1	20.7
30	11.35	16,394	12,564	3,831	3.3	0.45	0.19	24.01	17.2	23.2
35	12.38	17,525	13,251	4,274	3.1	0.43	0.18	22.91	18.3	24.6
40	13.42	19,700	14,818	4,882	3.0	0.41	0.17	21.82	19.9	26.9
45	14.45	21,436	15,990	5,446	2.9	0.39	0.16	20.83	21.1	28.7
50	15.48	36,706	29,145	7,561	3.9	0.37	0.16	19.77	27.7	38.4
55	16.51	38,242	30,161	8,081	3.7	0.36	0.15	19.18	28.8	39.7
60	17.54	46,110	36,888	9,222	4.0	0.35	0.15	18.66	32.0	43.9
65	18.58	47,299	37,574	9,725	3.9	0.34	0.14	18.14	32.8	44.9
70	19.61	49,478	39,165	10,313	3.8	0.33	0.14	17.66	33.9	46.1
75	20.64	58,648	47,095	11,552	4.1	0.32	0.13	17.14	37.0	49.4

Notes: **1.** Base Case Whittle shell prices used for design were 2.75/lb Cu, and $1,250/oz Au. **2.** NSR cut-off value of $10.32/t. **3.** Pit optimization used only Measured and Indicated Mineral Resources. **4.** Numbers may vary due to rounding.

Table 15-8: Ivy Ann Pit Optimization Results, August 31, 2018

Pit Shell	NSR Cut-off ($/t)	Total Tonnes (kt)	Waste Tonnes (kt)	Ore Tonnes (kt)	Strip Ratio (w:o)	Cu Grade (%Cu)	Au Grade (g/t)	NSR[2] Value ($/t)	Cu Tonnes (kt)	Au Ounces (oz '000s)
1	8.92	1,139	820	319	2.6	0.50	0.120	24.09	1.6	1.2
2	9.15	1,229	883	346	2.6	0.50	0.120	23.99	1.7	1.3
7	10.30	1,777	1,257	520	2.4	0.48	0.113	23.00	2.5	1.9
12	11.44	7,335	5,370	1,964	2.7	0.44	0.095	21.16	8.7	6.0
17	12.58	8,885	6,364	2,520	2.5	0.42	0.090	20.06	10.6	7.3
22	13.73	13,719	9,776	3,943	2.5	0.39	0.083	18.50	15.3	10.6
27	14.87	15,290	10,685	4,605	2.3	0.37	0.080	17.72	17.1	11.8
32	16.02	17,149	11,817	5,332	2.2	0.36	0.076	16.99	19.1	13.1
37	17.16	18,132	12,330	5,802	2.1	0.35	0.074	16.49	20.1	13.9
42	18.30	20,284	13,779	6,505	2.1	0.33	0.072	15.95	21.8	15.1
47	19.45	22,098	14,964	7,134	2.1	0.32	0.070	15.46	23.1	16.2
52	20.59	24,485	16,689	7,796	2.1	0.32	0.068	15.06	24.6	17.1
57	21.74	26,309	17,963	8,346	2.2	0.31	0.067	14.72	25.8	17.9
62	22.88	26,922	18,275	8,647	2.1	0.30	0.065	14.48	26.3	18.2

Notes: **1.** Base Case Whittle shell prices used for design were 2.75/lb Cu, and $1,250/oz Au. **2.** NSR cut-off value of $11.44/t. **3.** Pit optimization used only Measured and Indicated Mineral Resources. **4.** Numbers may vary due to rounding.

Table 15-9: Blackard Optimization Results, January 31, 2020

Pit Shell	NSR Cut-off ($/t)	Total Tonnes (kt)	Waste Tonnes (kt)	Ore Tonnes (kt)	Strip Ratio (w:o)	Cu Grade (%Cu)	Au Grade (g/t)	NSR2 Value ($/t)	Cu Tonnes (kt)	Au Ounces (oz '000s)
1	5.05	7,092	5,352	1,740	3.1	0.90	0	30.89	16	0
7	6.17	17,076	11,729	5,348	2.2	0.77	0	25.39	41	0
8	6.36	17,881	12,193	5,688	2.1	0.76	0	25.07	43	0
9	6.55	30,305	20,829	9,475	2.2	0.72	0	23.74	68	0
10	6.73	31,548	21,440	10,108	2.1	0.70	0	23.39	71	0
11	6.92	32,456	21,808	10,648	2.0	0.69	0	23.07	74	0
12	7.11	38,354	25,618	12,735	2.0	0.67	0	22.50	85	0
17	8.04	67,672	45,615	22,057	2.1	0.61	0	21.01	134	0
22	8.98	127,270	91,428	35,842	2.6	0.58	0	20.09	206	0
23	9.16	159,071	117,114	41,958	2.8	0.57	0	20.06	240	0
24	9.35	163,082	119,719	43,363	2.8	0.57	0	19.88	246	0
25	9.54	166,419	121,805	44,614	2.7	0.56	0	19.73	251	0
26	9.72	171,925	125,609	46,316	2.7	0.56	0	19.56	258	0
31	10.66	222,818	163,886	58,932	2.8	0.53	0	18.78	313	0
41	12.53	247,244	178,578	68,666	2.6	0.50	0	17.76	343	0
51	14.40	282,176	203,171	79,006	2.6	0.47	0	16.89	372	0
61	16.27	308,357	221,594	86,763	2.6	0.45	0	16.24	391	0
71	18.14	321,663	229,693	91,970	2.5	0.44	0	15.76	402	0
72	18.33	323,310	230,647	92,663	2.5	0.43	0	15.70	403	0
73	18.51	324,263	231,238	93,024	2.5	0.43	0	15.67	404	0
74	18.70	324,527	231,102	93,425	2.5	0.43	0	15.62	404	0

Notes: **1.** Base Case Whittle shell prices used for design were 2.75/lb Cu, and $1,250/oz Au. **2.** NSR cut-off value of $9.35/t. **3.** Pit optimization used only Measured and Indicated Mineral Resources. **4.** Numbers may vary due to rounding.

Table 15-10: Scanlan Optimization Results, January 31, 2020

Pit Shell	NSR Cut-off ($/t)	Total Tonnes (kt)	Waste Tonnes (kt)	Ore Tonnes (kt)	Strip Ratio (w:o)	Cu Grade (%Cu)	Au Grade (g/t)	NSR2 Value ($/t)	Cu Tonnes (kt)	Au Ounces (oz '000s)
1	6.40	3,402	2,569	833	3.1	0.91	0	28.04	8	0
7	7.64	15,229	10,549	4,680	2.3	0.80	0	24.79	37	0
12	8.67	23,020	15,856	7,164	2.2	0.74	0	23.14	53	0
17	9.70	28,030	19,152	8,878	2.2	0.70	0	22.07	62	0
18	9.91	30,262	20,759	9,502	2.2	0.69	0	21.78	66	0
19	10.11	36,341	25,374	10,967	2.3	0.67	0	21.30	74	0
20	10.32	38,103	26,681	11,422	2.3	0.66	0	21.14	76	0
21	10.53	38,953	27,218	11,735	2.3	0.66	0	20.98	77	0
22	10.73	39,639	27,642	11,997	2.3	0.65	0	20.84	79	0
27	11.76	53,438	37,877	15,561	2.4	0.62	0	19.71	96	0
32	12.80	59,020	41,818	17,202	2.4	0.60	0	19.16	103	0
37	13.83	61,939	43,602	18,337	2.4	0.58	0	18.69	106	0
42	14.86	79,377	58,302	21,075	2.8	0.57	0	18.36	120	0
47	15.89	83,548	61,256	22,292	2.7	0.56	0	17.96	124	0
57	17.96	94,716	69,593	25,123	2.8	0.53	0	17.15	133	0
67	20.02	103,877	75,948	27,929	2.7	0.50	0	16.31	141	0
68	20.23	105,583	76,888	28,695	2.7	0.50	0	16.07	142	0
69	20.43	105,772	76,975	28,797	2.7	0.49	0	16.04	142	0
70	20.64	106,230	77,261	28,969	2.7	0.49	0	15.99	143	0

Notes: **1.** Base Case Whittle shell prices for design were 2.75/lb Cu, and $1,250/oz Au. **2.** NSR cut-off value of $10.32/t. **3.** Pit optimization used only Measured and Indicated Mineral Resources. **4.** Numbers may vary due to rounding.

As noted in Table 15-4 through Table 15-10, the Eva Copper Project pit optimization results (pre-final design) show that, at prices of $2.75/lb for copper and $1,250/oz for gold, there would be 166 Mt of ore with average grades of 0.47% Cu and 0.05 g/t Au, and waste of 293 Mt.

The ultimate design pits obtained 171 Mt of ore grading 0.46% Cu and 0.05 g/t Au, and waste of 375 Mt, whereas the mining schedule had 170 Mt of ore and 374 Mt of waste, which is very close. The additional ore and waste tonnages are a result of incorporating bench and road designs, which generally adds more material. Re-evaluating and improving the mine designs is an ongoing process. CMMC anticipates lowering the strip ratios for all the Eva Copper pits.

16 MINING METHODS

16.1 Summary

The Eva Copper Project is anticipated to mine 170 million tonnes (Mt) of ore and 381 Mt of waste from seven open pit deposits, with a minimum projected mine life of 15 years. Conventional open pit mining methods will be employed at the Eva Copper Project, which includes drilling, blasting, loading, and hauling. The Eva Copper Project is estimated to have a one-year mine pre-production period. Mining activities are based upon open pit mining of the Little Eva deposit, initially at a rate of 31.2 kt/a of ore. The primary pits of Little Eva and Blackard will be supplemented by progressively mining seven smaller satellite pit areas at Turkey Creek, Bedford North and Bedford South, Lady Clayre, Ivy Ann, and Scanlan North and Scanlan South.

Diligent grade control will be a requirement throughout mining to maximize the grade of the material processed. The lower-grade, marginal ore from the Little Eva, Blackard, Scanlan, and Turkey Creek deposits will be stockpiled adjacent to the ROM pad and processed towards the end of mine life. Marginal mineralized material from the other satellite deposits has been classified as waste, and will not be transported to the process plant. The mill is to be located adjacent to the Little Eva and Turkey Creek pits, as they contain 55% of the mineral inventory.

Current estimated pit statistics and mine design parameters for the Little Eva, Turkey Creek, Bedford, Lady Clayre, Ivy Ann, Blackard, and Scanlan deposits are listed in Table 16-1. Surface rights are sufficient for the mine waste stockpile, tailings storage facility (TSF), and processing plant sites. An overall site plan for the Eva Copper Project is shown in Figure 16-1. A more detailed site plan of the Little Eva and Turkey Creek deposits and the processing facilities is shown in Figure 16-2.

The mining roster will be composed of four crews, who will alternate working 12-hour shifts over a two-week period. Operations are scheduled to be 24 h/d, 365 d/a scheduled time, which is common practice in Australia. Typical worked hours per month per shift employee are estimated at 183 hours. Overtime is estimated at approximately 11%.

Pit dewatering will be carried out from in-pit sumps and perimeter wells. It is expected that horizontal depressurization holes may be required as pit slopes develop in subsequent production years.

Initial shovel and truck availabilities are estimated to be 85% and 92%, respectively, which are reasonable. Equipment life hours used are based on those experienced at Copper Mountain Mining Corp. (CMMC)'s Canadian operations. For a 16-year mine life, capital for major rebuilds and/or replacements will be required for some of the mine equipment.

The Eva Copper Project grade control program will utilize blasthole sampling as its primary method of grade control.

Table 16-1: Eva Mine Design Parameters by Deposit

Input Factor	Unit	Little Eva	Turkey Creek	Bedford North/South	Lady Clayre	Ivy Ann	Blackard	Scanlan
Pit Dimensions								
Longitudinal Axis (length)	m	1,700	1,100	630/340	730	490	1,900	740
Transverse Axis (width)	m	950	400	260/200	350	350	660	465
Vertical Axis (depth)	m	310	170	110/80	100	120	240	220
Pit Bottom Elevation	mASL	-140	-10	80	100	80	-50	-30
Top Bench Elevation	mASL	170	170	190	215	200	200	200
Maximum Pit Depth	m	280	180	110	115	120	250	230
Average Depth of Weathered Profile	m	15-to 25-m	South: 20-to 30-m	20-to 30-m	15-to 30-m	20-to 30-m	30-to 50-m	20-to 25-m
Pit Dewatering Needed	Yes/No	Yes	Yes	Yes	Yes	Yes	Yes	Yes
Current, Estimated Pit Dewatering	m^3/d	4,000	TBD	TBD	TBD	TBD	TBD	TBD
Approximate Distance to Nearest Waste Dump	m	750	750	500	500	500	500	500
Elevation of Nearest Waste Dump	m	750	750	500	500	500	3	2
Number of Pit Phases	num.	6	2	1	1	1	3	1
Approximate Distance to ROM pad	m	1,000	1,000	6,900	19,700	35,850	6,800	18,800
Overall Hanging Wall Slope	deg.	56	43	43	43	43	43	43
Overall Footwall Slope	deg.	43	43	43	43	43	43	43
Minimum Mining Width	m	60	60	60	60	60	60	60
Average Berm Width – Oxide	m	10	10	10	10	10	10 to 14	10
Average Berm Width – Sulphide		10	10	10	10	10	10 to 14	10
Average Bench Face Angle	deg.	80	80	80	80	80	80	80
Average Sinking Rate (max)	bench/a	12	12	12	12	12	12	12
Average Ramp Width	m	30	30	30	30	30	30	30
Average Ramp Grade	%	10%	10%	10%	10%	10%	10%	10%

Input Factor	Unit	Little Eva	Turkey Creek	Bedford North/South	Lady Clayre	Ivy Ann	Blackard	Scanlan
Type of Benching	num.	Single/Double	Single/Double	Single	Single	Single	Double	Single
Drilling and Blasting								
Final Bench Height	m	10	10	10	10	10	10	10
Sub-drill Height	m	1.4	1.4	1.4	1.4	1.4	1.4	1.4
Average Blasthole Spacing – Ore	m	6.3	6.3	6.3	6.3	6.3	6.3	6.3
Average Blasthole Spacing – Waste	m	6.9	6.9	6.9	6.9	6.9	6.9	6.9
Average Stemming Height	m	4.5	4.5	4.5	4.5	4.5	4.5	4.5
Blasthole Diameter	mm	250/127	250/127	250/127	250/127	250/127	250/127	250/127
Average Holes per Blast	num.	200	100	100	100	100	200	150
Average Drill Penetration Rate	m/min op	0.36	0.36	0.36	0.36	0.36	0.36	0.36
Average Shovel Production Rate	t/h op	2,417	2,417	2,417	2,417	2,417	2,417	2,417
Average Truck Production Rate	t/h op	409	409	409	409	409	409	409
Bucket Fill Factor	%	95%	95%	95%	95%	95%	95%	95%
Truck Fill Factor	%	98%	98%	98%	98%	98%	98%	98%
Operating Efficiency	%	83%	83%	83%	83%	83%	83%	83%
Scheduled Hours Per Shift	h/shift	12	12	12	12	12	12	12
Shifts	Shift/d	2	2	2	2	2	2	2
Crews	num.	4	4	4	4	4	4	4

The Eva Copper Project was evaluated using a range of shovel and truck sizes, and a 15 m^3 to 22 m^3 shovel, matched with 141 tonne class trucks, as it was determined to best balance waste movement efficiency and ore selectivity, equipment prices, resource deposit characteristics, and grade control selectivity. This fleet will also be used for mining the satellite deposits of Turkey Creek, Bedford, Lady Clayre, Ivy Ann, Blackard, and Scanlan. Hauling of ore from Ivy Ann to the processing plant will probably be performed by a contractor, which has been assumed in the mine costing.

CMMC has followed the blasting powder factors for the mine as recommended by George Orr and Associates; both ore and waste at approximately 0.25 kg/t, or the fresh rock blasting to fracture at the middle of the recommended range of 0.6 kg per bank cubic metre (bcm) to 0.8 kg/bcm. The first few months of mining in Little Eva, most of the material mined will be weathered material; however, most of the ore material mined in the LOM plan will be fresh rock. Estimates are 6.3 m for blasting burden, 6.9 m for blasthole spacing, and 1.4 m for sub-drill for a 10 m bench height blasted. Average blasthole stemming is estimated at 4.5 m.

Eva's LOM strip ratio for the design pits is relatively low to moderate, at approximately 2.2:1 waste to ore (w:o). Mined tonnage is proposed to be at a relatively constant rate throughout the LOM. The mine requires approximately 13.52 Mt of pre-stripping in the first year of mining, which is the final pre-production year. It should be noted that a large part of this material will be used to build the initial TSF, some roads, and part of the Cabbage Tree Creek bund.

Total waste material movement is estimated to be approximately 67,000 t/d, and direct ore haulage is estimated to be 31,200 t/d (11.4 Mt/a). Waste will be hauled to designated waste dumps specifically designed for this purpose. Some of the Little Eva, Turkey Creek, Bedford, and Blackard waste material will be used for continued construction of the TSF, Cabbage Tree Creek diversion bunding, and other site infrastructure.

The Cabbage Tree Creek diversion channel and bund will be located along the western and northern sides of the Little Eva pit, and is required to divert water travelling along the Cabbage Tree Creek. This will be constructed early in development to ensure there is no water inundation of the pit. It should also be noted that water bund, water diversion, and sediment containment structures will also be built for the other open pits and waste dumps.

The starter TSF will be constructed to be available for the commencement of operations of the process plant. Preparation of the TSF basin and embankment foundations will be carried out by CMMC mining crews with support from a contractor prior to approximately 1 Mm^3 of mine waste being transported, conditioned, and placed into TSF embankment construction by Eva personnel. The starter dam of the TSF will provide one year of tailings capacity (approximately 11.4 Mt). Over the LOM, the TSF embankments will be continuously raised using Little Eva, Blackard, Bedford, and Turkey Creek mine waste delivered and placed by the mining crews to ensure adequate tailings storage capacity is available to facilitate mill production. Haul distances to the TSF will vary from 1.5 km to 2.0 km from the Little Eva, Turkey Creek, and Blackard pits. The TSF design at the final stage of construction will have a storage capacity of 170 Mt, with the highest embankment height above natural ground of 52 m (KCB, 2019), and a total mine waste installed embankment volume of approximately 26.0 Mm^3. The TSF basin area is estimated to be approximately 424 ha. Using waste from multiple pit areas is partially dependent on the dam location distance to the nearest waste source, e.g., on the south end of the TSF, it would be less expensive to haul waste from Blackard than Little Eva.

Weather and altitude should not impact productivities; however, severe rainfall can occur from December through March. Drought may impact the water supply, as the Eva Copper Project will primarily rely on make-up water from the tailings dam, and relatively deep water-wells.

The current mine site layout is shown in Figure 16-2.

16.2 Waste Rock Storage Facilities

Haul distances to the waste rock storage facilities and run-of-mine (ROM) ore stockpile crusher area are moderate (approximately 0.5 km to 2.0 km in the first five years). Table 16-1 shows the relative distances from the different deposits to the primary crusher. Eva Copper Project consultants and personnel are evaluating alternate waste dump locations. If acceptable, alternative dump locations have been identified for the latest mining permit application. Table 16-2 summarizes the waste dump parameters used for design, and the resulting capacities for all five deposit areas. All dumps have been designed to accept an additional 5% to 10% more material. It should be noted that approximately 65 Mt of waste material will be needed to build the TSF and Cabbage Tree Creek (CTC) bund. This material will be sourced from the Little Eva, Turkey Creek, and Blackard pits. Bedford will also be a source of waste rock for the southwest corner of the TSF.

Table 16-2: Eva Waste Rock Storage Facilities Design Criteria, 2019

Input Factor	Unit	Little Eva	Turkey Creek	Bedford	Lady Clayre	Ivy Ann	Blackard	Scanlan
Road Grade	%	10%	10%	10%	10%	10%	10%	10%
Minimum Road Width	m	22	22	22	22	12	22	22
Catch Bench Width	m	6	6	6	6	6	6	6
Swell Factor	num.	0.714	0.714	0.714	0.714	0.714	0.714	0.714
Loose Density	t/m^3	2.01	2.01	2.01	2.01	2.01	2.01	2.01
Angle of Repose	H:V	1.5:1	1.5:1	1.5:1	1.5:1	1.5:1	1.5:1	1.5:1
Lift Slope	deg.	33.7	33.7	33.7	33.7	33.7	33.7	33.7
Overall Slope	deg.	25.5	25.5	25.5	25.5	25.5	25.5	25.5
Final Slope	deg.	18.4	18.4	18.4	18.4	18.4	18.4	18.4
Typical Lift Dump Height	m	10	10	10	10	10	10	10
Capacity	Mt	169.2	Combined with Little Eva	18.2	17.7	9.9	139.4	29.0

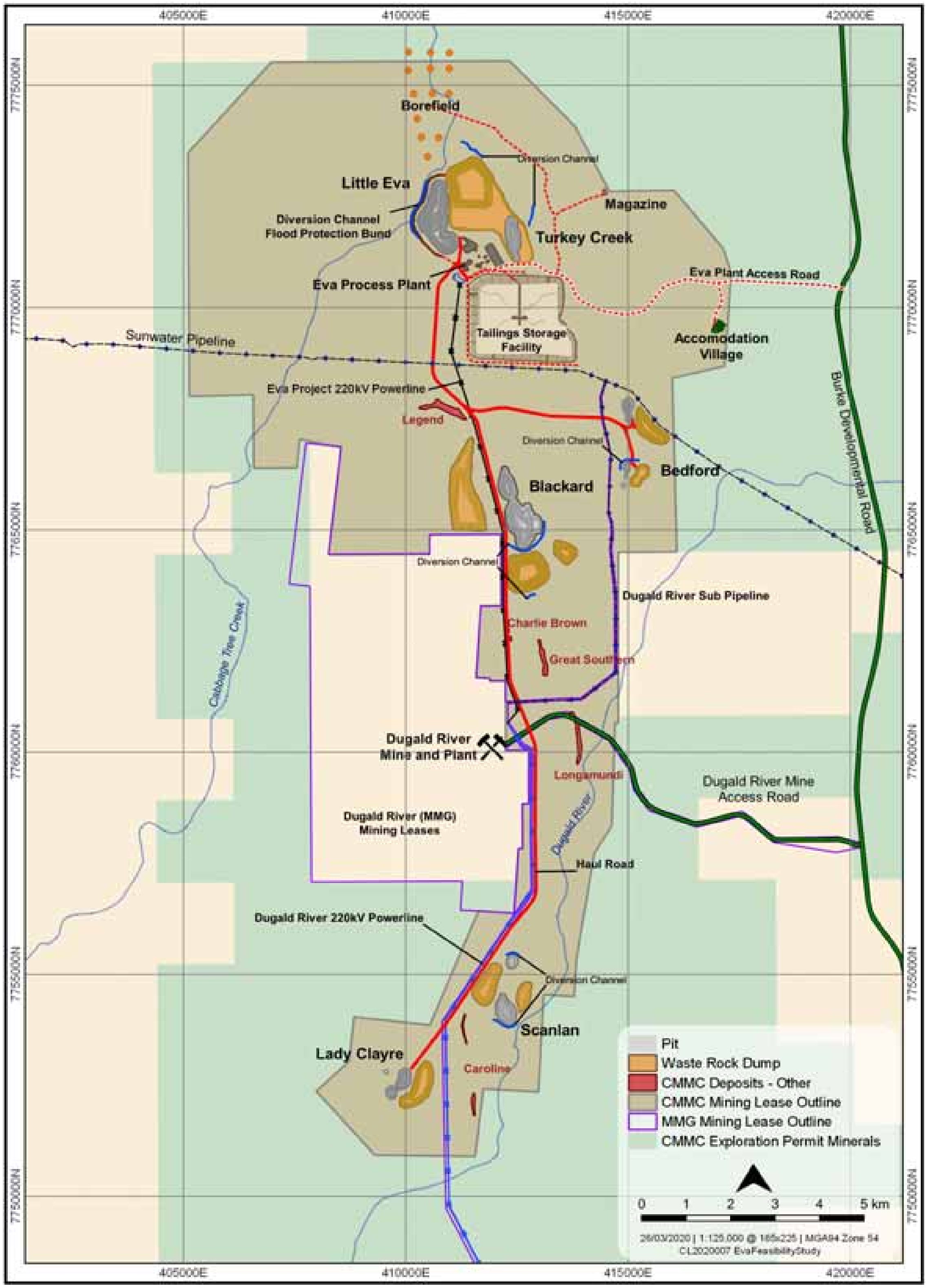


Figure 16-1: Eva Copper Project Site Layout

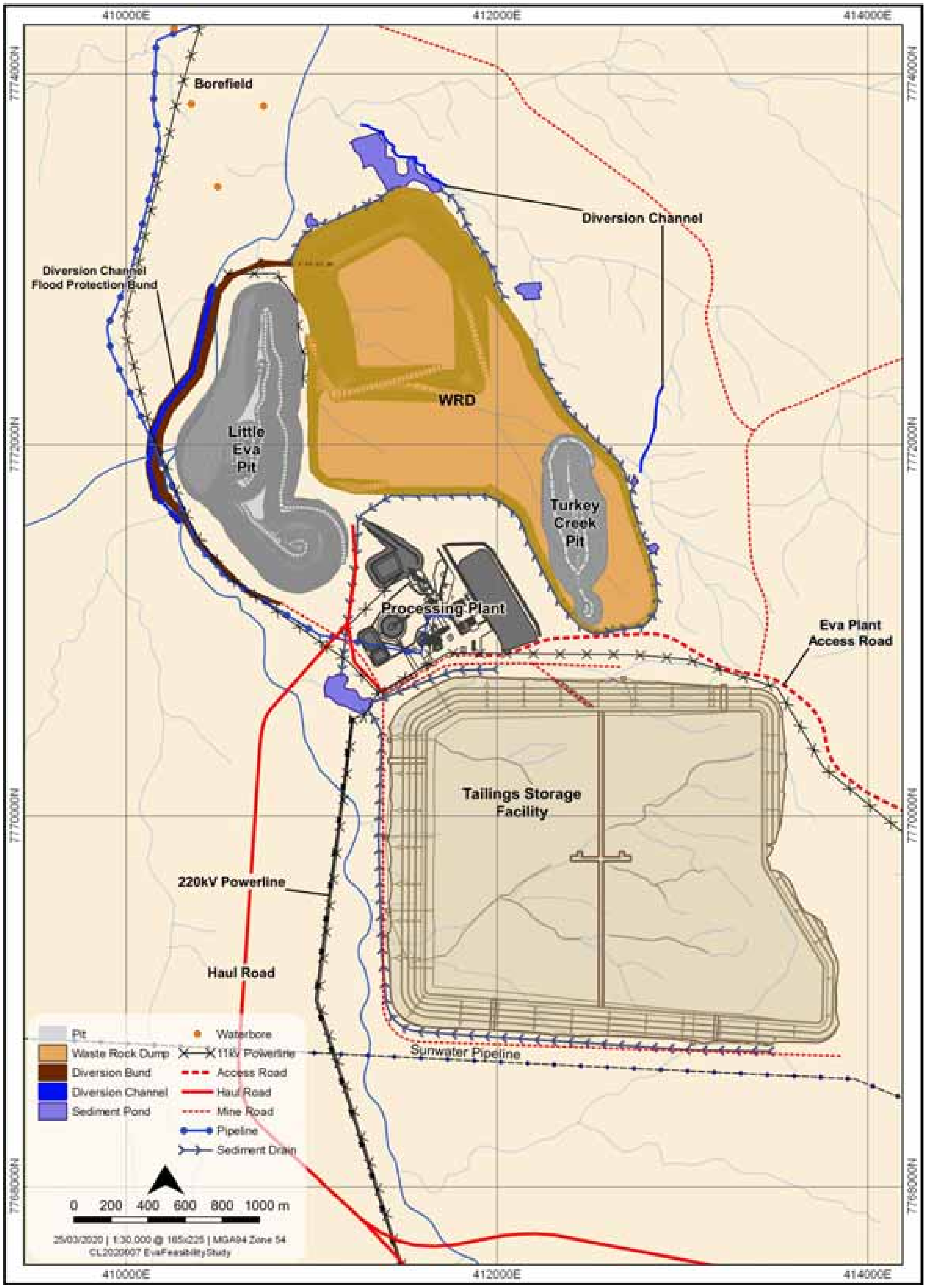


Figure 16-2: Eva Copper Project – Little Eva, Turkey Creek, and Process Plant

16.3 Geomechanics

Geotechnical studies have been carried out on the Little Eva, Blackard, and Scanlan deposits to provide pit design and mining inputs.

Other than geotechnical data collected from a 2015 diamond drilling campaign at the Turkey Creek deposit, there have been no formal geotechnical programs carried out on any of the other satellite deposits to date; however, the geotechnical review of the satellite pits suggest that the slopes chosen are conservative.

For the purposes of generating mining operating costs, it has been assumed that the geotechnical conditions in the satellite deposits are similar to those generally found in the Little Eva, Blackard, and Scanlan deposits. Drill rates, powder factors, slope angles, bench heights, and truck and shovel mining methodologies have been assumed standard across all deposits.

The Little Eva deposit has had two geotechnical studies completed, the first as part of the 2005 DFS, and the second in 2012, which formed the basis for the 2012 DFS pit design. These studies were completed by George Orr and Associates. The 2012 DFS pit design remains the design used in the 2014 DFS update and the parameters were used in this update. Blackard and Scanlan had a geotechnical evaluation for mining feasibility purposes completed in June 2006 by George Orr and Associates.

An improved, integrated geological-geotechnical model was completed by Altona in 2014 for Little Eva that supports the previous Little Eva studies.

16.3.1 Little Eva Geomechanics

George Orr and Associates conducted a full stability analysis of the planned Little Eva pit based on geotechnical analysis of 21 oriented diamond drill holes (DDH) covering both an earlier starter pit design and the final pit design utilized in this study. The northwest portion of the deposit has poor to moderate ground conditions; however, for most of the planned pit, ground conditions are good to moderate. Overall slope angles of 43 degrees, inclusive of pit ramps, have been recommended, and are used in the Little Eva pit design. The eastern pit wall has the best ground conditions, and therefore all access ramps have been placed on this wall.

A simple model of the broad geotechnical character of the deposit has been developed from integrating the new geological model with the rock quality designation (RQD) and uniaxial compressive strength (UCS) data from these studies. The geological-geotechnical model defines eight distinct rock quality subdomains, as summarized in Table 16-3. The broken zone along the northwestern edge of the deposit is the only major zone of weakness identified by the model, and the principal source of risk in the pit design.

Slope design methodology by George Orr and Associates included kinematic stability assessments and two-dimensional limit equilibrium stability analyses against potential wall failure. The geological model used is the Universal model from 2005, updated for the 2011 Mineral Resource model, in which the distribution of weak rock and the location and geometry of the Coolullah fault were poorly constrained. The projected coincidence of the Coolullah fault and weak rocks on the western margin of the deposit, and the uncertainty around their location and limits, led to the adoption of a conservative approach to pit design. The 2014 study and the geotechnical model in this study demonstrated that this interpretation was too conservative, and there may be an opportunity to improve the design.

Table 16-3 and Table 16-4 summarize the geotechnical mine design parameters that were used in the 2019 CMMC Eva Copper Project Feasibility Study. It was recommended that no haul roads be placed on the western wall, and that an overall pit slope angle of 43 degrees be applied. A 56-degree slope angle was recommended for the remainder of the pit, dropping to 43 degrees when inclusive of ramps in the eastern wall. The west wall is to have 10 m vertical height faces, 9 m berms, and 80-degree face angles. The east, north, and south walls are to have either 10 m vertical height faces, 5 m berms, and 80-degree face angles, or 20 m vertical height faces, 10 m wide berms, and 80-degree face angles.

Given further and better definition of the “broken zone” on the western edge of the deposit, its intersection with pit designs can be better understood, and there may be opportunity to improve pit designs in this area in the future.

It is assumed that all walls will be substantively dewatered and depressurized, which will lessen the impacts of water on slope stability.

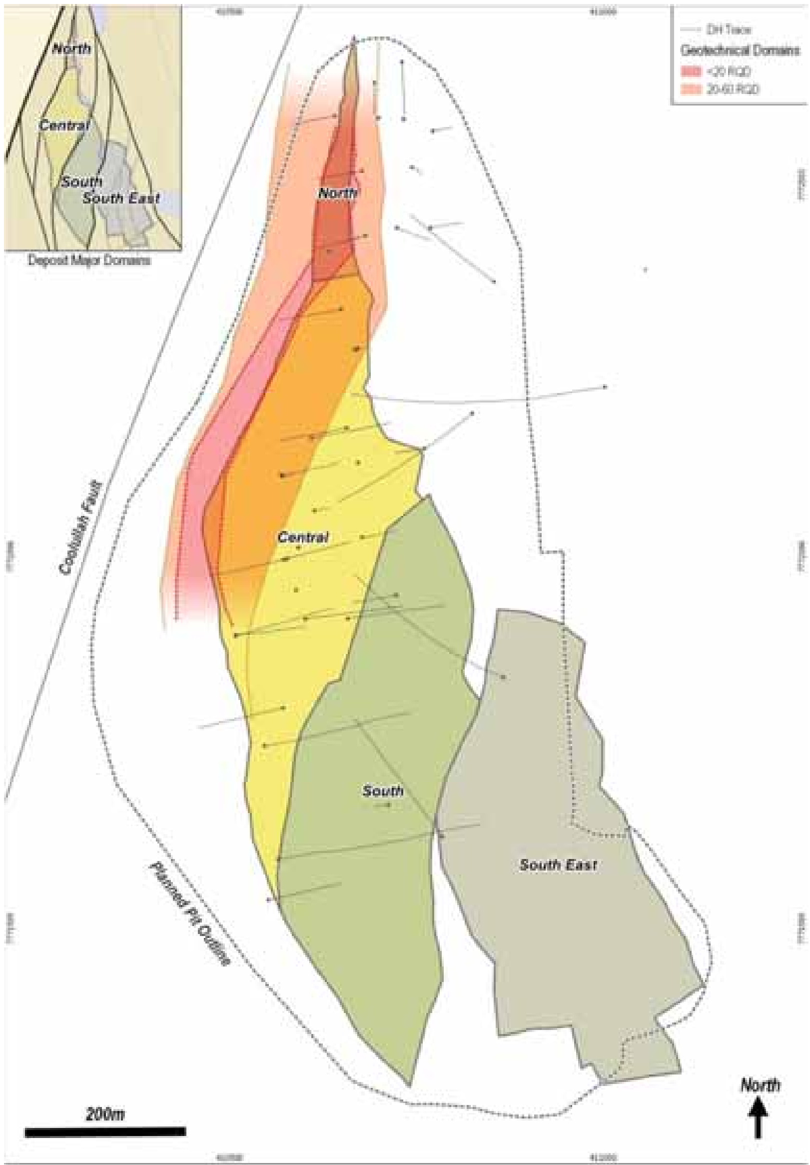
The weathered rock zone varies from 5 m to 25 m in depth, and it was recommended that within this zone walls be mined at a pit slope angle of 40 degrees with 10 m vertical height faces, 5 m wide berms, and 55-degree face angles.

Excavation requirements were guided by the geotechnical analysis of intact rock strength and rock defect spacing.

Table 16-3: Little Eva Mine Design Geotechnical Domains, 2018

Geological Domains	Rock Quality Subdomain	Comment
Oxidized Rocks		
Cover	-	Typically, thin colluvium or residual soils to approximately <3-m depth Alluvium in creek channel along northern half of western pit edge to approximately 25 m depth with potential shallow aquifers
Metasediments	-	Typical depth of base of weathering is 5 m to 25 m. Majority of the upper part of final pit wall in this material
All Igneous Domains	-	Typical depth of base of weathering is 5 to 25 m
Fresh Rocks		
Metasediments External Wallrocks (deposit hanging wall and footwall)	‘C’ - Competent	Interlayered fine to medium-grained schists, carbonates, and calc-silicate rocks Layering/fabrics interpreted to be largely subparallel to the igneous domain contacts
	‘MB’ - Moderately Broken	RQD values 20% to 60% with occasional broken ground
	‘B’ - Broken	Broken or RQD values < 20% throughout
North Resource Domain	‘B’ - Broken	Broken or RQD values < 20% throughout Dominated by hydrothermal brecciated altered intermediate igneous rocks

Geological Domains	Rock Quality Subdomain	Comment
Central Resource Domain	MB' - Moderately Broken	RQD values of 20% to 60% with occasional broken ground. Consistently broken along the western metasediment contact Heterogeneous, with subdomains of hydrothermal breccia and altered intermediate and felsic (10%) igneous rocks
	'C' - Competent	Heterogeneous, with subdomains of hydrothermal breccia and altered intermediate and felsic igneous rocks
South Resource Domain	'C' - Competent	Dominated by broad zones of altered intermediate igneous rocks Extensive areas of internal waste
South East Resource Domain	'C' - Competent	Rocks similar to South domain and interpreted to have similar properties Extensive areas of internal waste



Source: Altona DFS, August 2017

Figure 16-3: Little Eva Geotechnical Domains

16.3.2 Blackard Geomechanics

George Orr and Associates evaluated the Blackard pit for a 130 m vertical wall height. Before mining commences at Blackard, their studies need to be updated for a 240 m vertical wall height. Overall, Blackard mine designs are similar to the recommendations of George Orr and Associates. Overall, slope heights used for the design average 40 degrees to 43 degrees. George Orr and Associates recommended a slope angle range from 38 degrees to 46 degrees, depending on the mining rock mass rating (MRMR) values. It must be stressed that dewatering and depressurization of the pit slopes is critical in maintaining stable pit slopes, as the water table is only 10 m below the surface of the pit. Depressurization of the pit slopes will need to be completed before mining is initiated. If the slopes are not depressurized, the chances of slope failure will increase.

Rocks at Blackard are weathered and altered. Mineralization occurs in an altered sandstone. The following BCD (Blackard deposit) core holes were tested and measured: 469, 470, 481, 493, 495, 496, 497, and 498. Several UCS tests, triaxial compression tests, and moisture content tests were performed on selected Blackard core samples. George Orr and Associates' interpretation of the intact rock strengths range from Very Weak (UCS values from 0.45 MPa to 1.25 MPa) to Moderately Strong (UCS values from 12.5 MPa to 50 MPa). Testing showed a large range of cohesion, from 171 kPa to 449 kPa, and friction angles from 32 degrees to 45 degrees. RQD values are generally low; however, values increase in the unaltered rock, which is a positive finding.

Structurally, principal bedding has a dip direction of 270 degrees and an approximate dip angle of 70 degrees, and principal joint orientations tend to be parallel to bedding orientations. There are more direction and dip orientations for both bedding and joints. Faulting was observed in all boreholes evaluated. The western boundary of the deposit is formed by a north-striking, sub-vertically dipping geological structure referred to as the Western Boundary Fault Zone (WBFZ). The central and eastern alteration boundaries have a shallower, westerly dip, which is locally offset by sub-vertical faulting, and it trends sub-parallel to the WBFZ. East- to northeast-striking faults have also been observed.

Toppling-type failure on the west wall above a pit depth of 130 m constitutes a major geotechnical risk at Blackard, according to George Orr and Associates. Dewatering the slopes in advance of mining, installing horizontal drain holes, using controlled blasting and digging practices, and conducting regular pit mapping and rock defect surveys will all aid in minimizing this risk. Small bench failures will be expected.

Further hydrogeological and geotechnical analyses of the Blackard pit are required before mining begins.

16.3.3 Scanlan Geomechanics

Scanlan's geomechanical characteristics are quite similar to Blackard; structurally complex, shallow water table (10 m to 17 m below surface); much of the pit walls will be mined in an altered sandstone; toppling failure on the west pit wall constitutes a major geotechnical risk; good quality blasting and digging practices will lessen the chances of any serious slope failures; and rocks can be altered and weathered. Designed pit slopes averaged 38 degrees to 43 degrees for the Scanlan pit. Because of the erratic nature of the altered rock zones at Blackard and Scanlan, the exact final slopes are to be determined after the pits are opened. Pit wall and floor mapping will be used as a guide to amend the pit slope angles, as necessary.

Overall, slope heights used for design average 43 degrees. George Orr and Associates recommended slope angles ranging from 38 degrees to 56 degrees, depending on the MRMR values. It must be stressed that dewatering and depressurization of the pit slopes is critical in maintaining stable pit slopes at Scanlan, as the water table is only 10 m to 17 m below the surface of the pit. Depressurization of the pit slopes will need to be completed before mining is initiated. If the slopes are not depressurized, the chances of slope failure will increase.

The following SCD (Scanlan deposit) core holes were tested, measured, and interpreted: 165, 166, 167, 168, 169, 170, and 171. Several UCS tests, triaxial compression tests, and moisture content tests were performed on selected Scanlan core samples. George Orr and Associates' interpretation of the intact rock strengths range from Very Weak (UCS values from 0.8 MPa to 3.8 MPa) to Moderately Strong (UCS values from 3.0 MPa to 88.8 MPa). Testing showed a large range of cohesion, from 392 kPa to 786 kPa, and friction angles from 29 degrees to 37 degrees. RQD values are generally low; however, values increase in the unaltered rock.

It appears that the principal bedding orientation is defined by a 65-degree dip angle, and a dip direction of 280 degrees. Joint orientations at Scanlan appear to be characterized by a southwest- to westerly-dipping “ring,” with dips around 45 degrees.

Wet blasting conditions may occur if the pit slopes are not adequately dewatered. It is anticipated that much of the altered rock at both Scanlan and Blackard could either be lightly blasted, ripped, or possibly even freely dug.

Further hydrogeological and geotechnical analyses of the Scanlan pits are required before mining begins. This Scanlan pit is deeper than the original design evaluated by George Orr and Associates in 2006.

16.3.4 Turkey Creek Geomechanics

The Turkey Creek pit will be 175 m deep. No formal geotechnical studies have been carried out to date.

Six DDHs were completed in 2015 to collect metallurgical samples. Five of the holes were drilled to test within the proposed pit shell, and extended into the footwall of the deposit through the pit wall. RQD and “broken ground” logging was carried out.

Fresh metasediments in both the footwall and hanging wall were consistently competent, with RQD greater than 80%. No major fractured or broken ground in structures were logged.

UCS measurements were completed at Australian Laboratory Services (ALS) on 15 representative metasedimentary core samples. The average UCS was 57.8 MPa, and UCS values ranged from 17.9 MPa to 123 MPa. The strength classification for the majority of the samples was Strong to Medium Strong, and the type of failure mode was determined to be by shear.

The overall pit slopes adopted were 56 degrees (43 degrees where inclusive of ramps). A formal geotechnical study utilizing oriented drill core will be undertaken before the final mine design and mining.

16.3.5 Bedford, Lady Clayre, and Ivy Ann Pits

All of these pits are relatively shallow, with Bedford at 110 m in depth, Lady Clayre at 115 m in depth, and Ivy Ann at 120 m in depth. Pit slope angles used for all three satellite pits are 43 degrees. Evidence suggests that there are no areas of significant broken ground in these satellite pits similar to the fractured zones found at the Little Eva pit. More formal geotechnical investigations will be performed prior to mining of these pits.

16.3.6 Blackard Pit

Blackard is similar in size to the Little Eva pit. Pit depth is approximately 235 m, and the pit slope angles used are 43 degrees. Evidence suggests that there are no areas of significant broken ground in this satellite pit similar to the fractured zones found at the Little Eva pit. More formal geotechnical investigations will be performed prior to mining of this pit.

16.3.7 Scanlan Pit

The Scanlan pit is moderate in size compared to the Little Eva and Blackard pits. There are currently two pits designed for this resource, Scanlan North and Scanlan South. The larger South pit depth is approximately 225 m, and the pit slope angles used are 43 degrees. Evidence suggests that there are no areas of significant broken ground in this satellite pit similar to the fractured zones found at the Little Eva pit. More formal geotechnical investigations will be performed prior to mining of this pit.

Table 16-4: Eva Copper Recommended Slope Design Parameters, 2019

GEMS/Whittle Rock Zone	Little Eva-West Side	Little Eva-East Side	Turkey Creek[1]	Bedford[1]	Lady Clayre[1]	Ivy Ann[1]	Blackard	Scanlan
Rock Zone	Single Bench	Double Bench	Single Bench	Single Bench	Single Bench	Single Bench	Double Bench	Single Bench
Overall Slope, (degrees)	43	52	43	43	43	43	40 - 43	38 - 43
Bench Height, m	10	20	10	5	5	5	10	10
Batter Angle, (degrees)	80	80	55 to 80	56 to 80	57 to 80	58 to 80	70	70
Catch- Bench Width, m	9	10	7 to 10	5	5	5	14	7
Angle, (degrees)	43	56	43	43	43	43	43	43
Wall Height, m	280	280	180	110	115	120	235	225
No. of Ramp Crossings (est.)	0	3	0	0	0	0	2	2
Ramp Width, m	0	22	22	22	22	22	22	22
Inter-Ramp Angle, (degrees)	43	45	43	43	43	43	43	43

Notes: [1] Berm (catch bench width) and batter values will vary depending on weathering profile, either oxide or fresh rock.

Pit slope safety will be the number one priority for CMMC mining crews in all pits. It is also recognized that high wall slope failures can cost the company in both lost production and increased waste mining.

Upon closure and pit reclamation, all of the pits will have the necessary bund structures (no closer than ten metres from the edge) built around the pit perimeters. As a guideline, the Western Australia-Department of Industry and Resources guidelines will be used to determine the final location. Fresh, unaltered rock is the preferred material for bund construction.

16.4 Mine Design

The mine planning is appropriate for this stage of production. One of the primary reasons for this level of engineering is the need to have accurate information for production, budgeting, and permits. Current economic models have been based on LG optimized pits using a varied number of slopes for the Little Eva and satellite pits. Strip ratios (w:o) are accurately presented in the ultimate pits and LOM schedules; LOM strip ratio is 2.2:1 (w:o).

All ultimate pits are based on designs that incorporate benching, haul roads, and waste storage facilities. Bench-by-bench, monthly production schedules for the first two years of mining have been developed to identify ore types, waste removal, and stripping requirements. A lower stripping ratio will be realized in the latter years.

Bench height is 10 m with 1.6 m of sub-drill for the Little Eva and Turkey Creek deposits, which comprise over 55% of the Mineral Reserves. Little Eva, Turkey Creek, Blackard, and Scanlan road widths are 22 m. Normal berm widths in the Little Eva and Turkey Creek pits are approximately 6 m to 10 m in the oxide, transition, and fresh rock, depending on each specific batter angle. Berm widths at Blackard were increased to 14 m.

Waste rock facility (waste dump) designs have been completed to a moderate level of detail. The mining engineers are evaluating the cost of locating additional waste dumps closer to the open pits, which should reduce the overall unit mining cost.

The current Little Eva pit design is approximately 1,700 m long and 950 m wide, with elevations that range from a high of 170 mASL to a low of -140 mASL. Initial pit exits for the south and north ramps are both at approximately 160 mASL. The crusher dump pocket is approximately 1 km horizontal distance from the Little Eva and Turkey Creek pit exits. The Eva Copper Project has left adequate buffers around the open pit for possible future expansions, should the price of copper increase.

Condemnation holes have been drilled in the infrastructure and waste dump areas.

Figure 16-4 shows the Little Eva ultimate pit with the extent of the designed mining limit. The process of obtaining all permits for the open pits, waste dumps, and tailings dam is in progress for the Eva Copper Project. Figure 16-4 through Figure 16-9 show plan maps of the open pit excavations and waste dumps, and topographies as of January 31, 2019 for the Little Eva, Turkey Creek, Bedford, Lady Clayre, Ivy Ann, Blackard, and Scanlan deposits.

Figure 16-11 is a typical cross-section of the proposed pit wall configuration for the Eva Copper Project pits. Safety is the number one concern for CMMC, and maintaining stable pit slopes is a primary mining objective for CMMC.

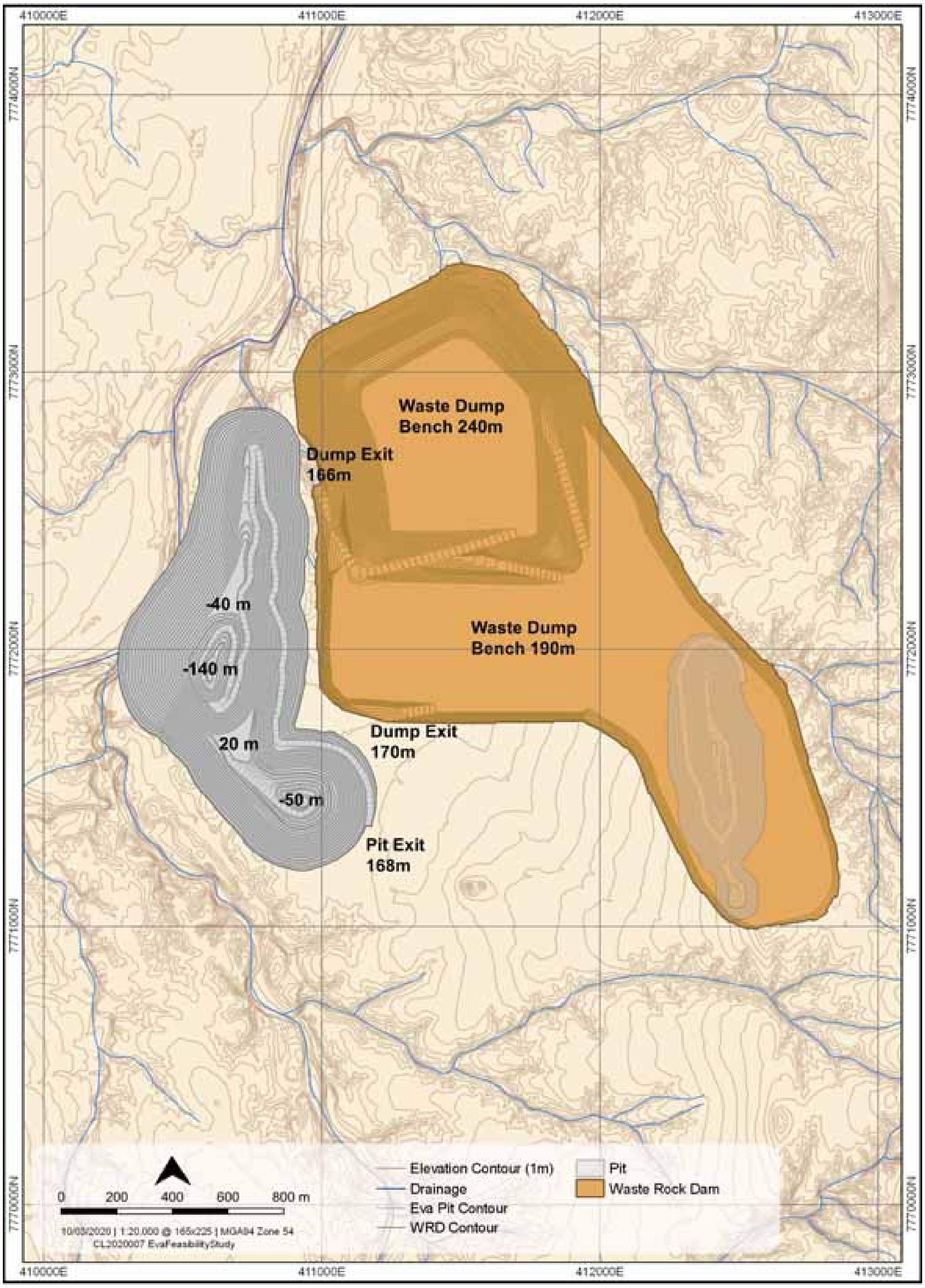


Figure 16-4: Little Eva Ultimate Pit and Waste Dumps, 2019

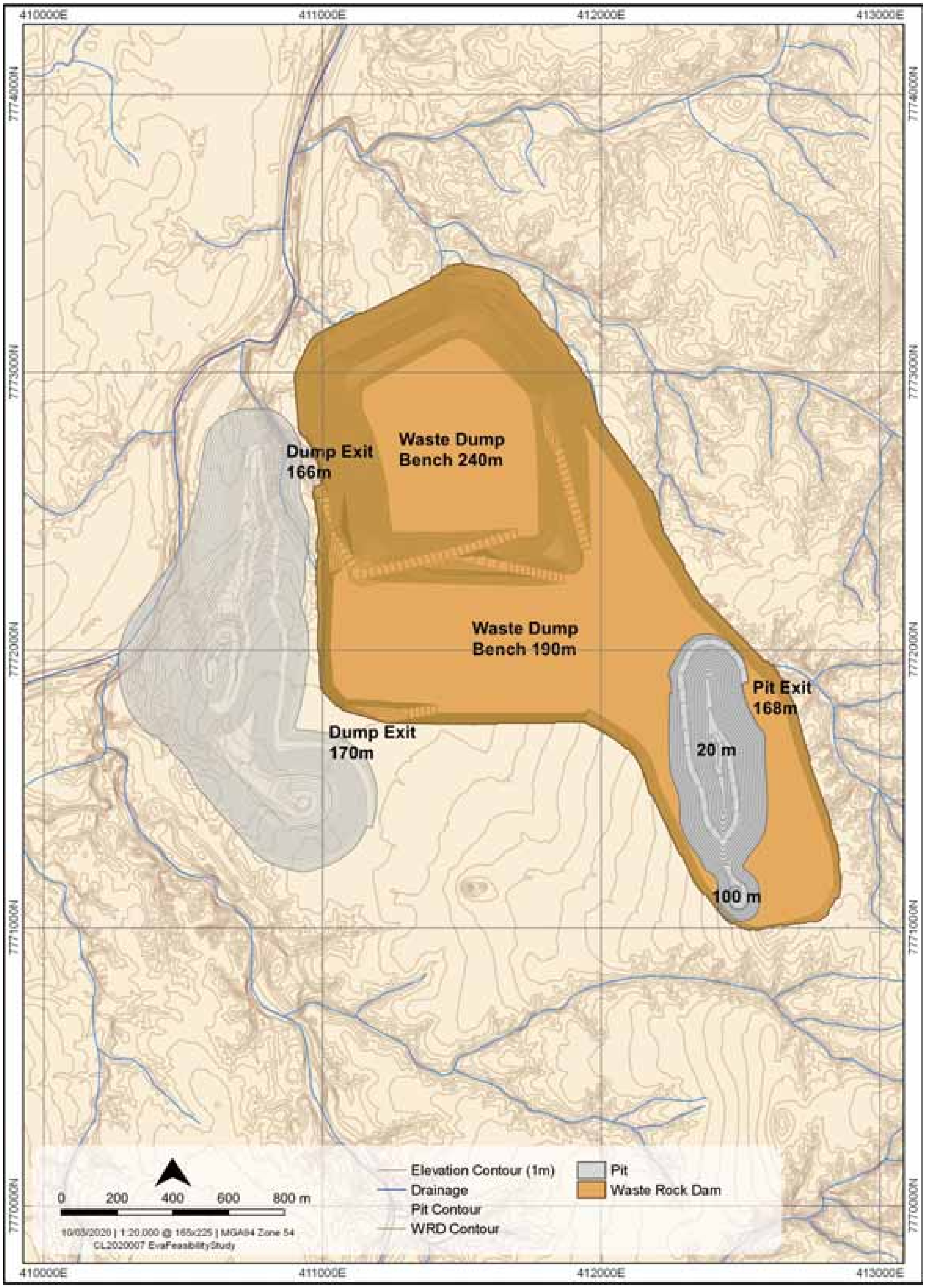


Figure 16-5: Turkey Creek Pit and Waste Dump, 2019

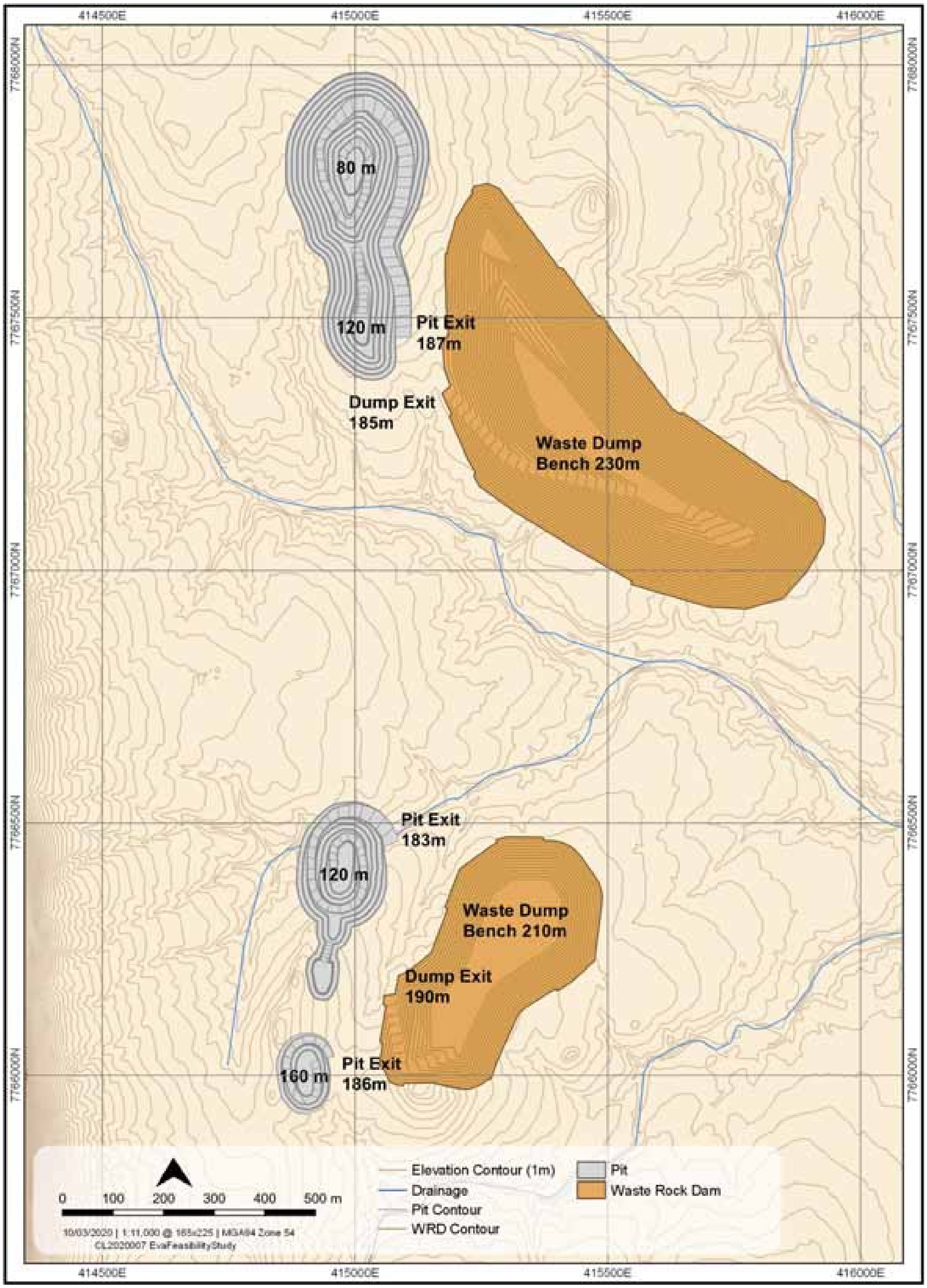


Figure 16-6: Bedford North and South Pits and Waste Dumps, 2019

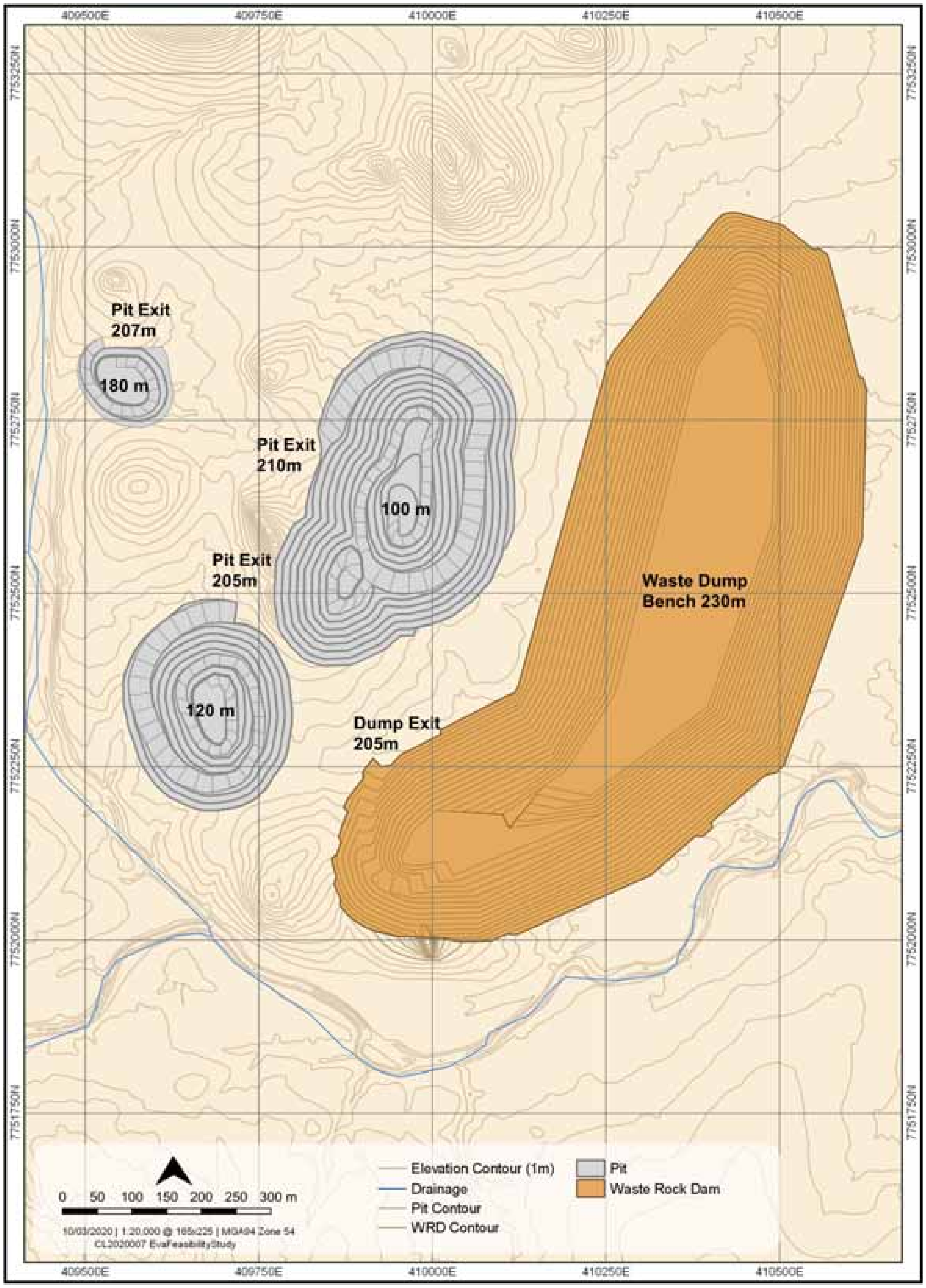


Figure 16-7: Lady Clayre Pit and Waste Dump, 2019

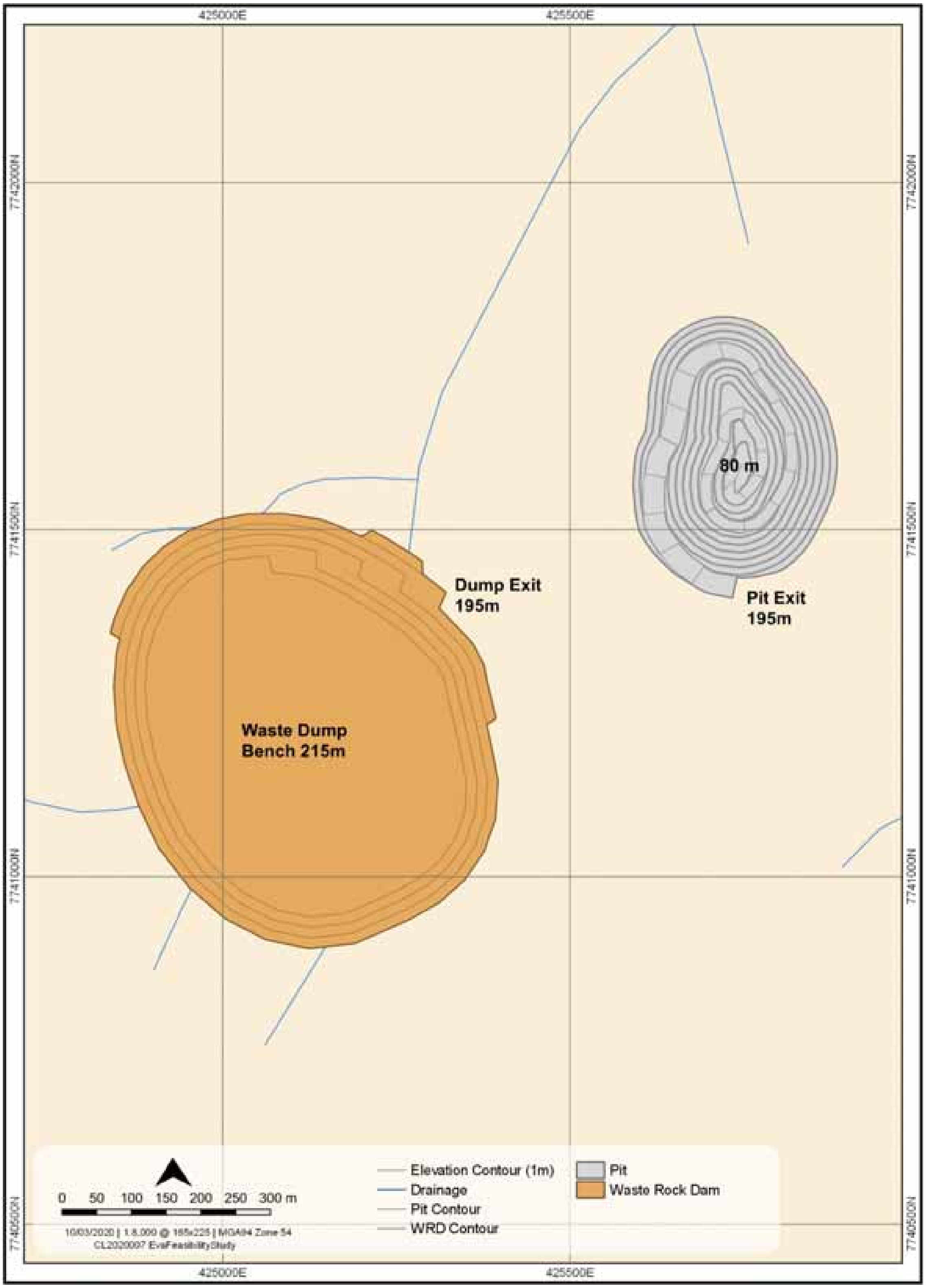


Figure 16-8: Ivy Ann Pit and Waste Dump, 2019

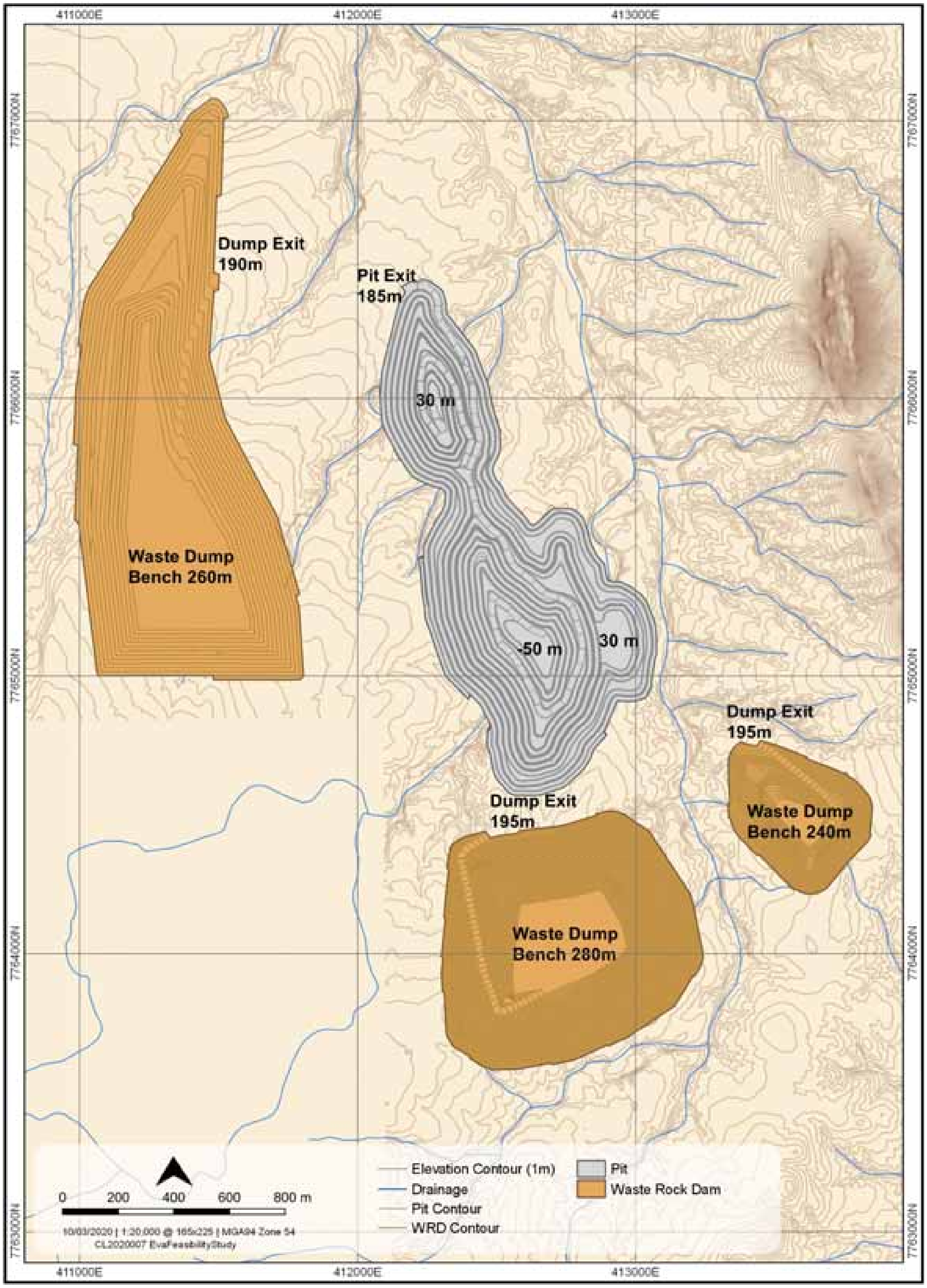


Figure 16-9: Blackard Pit and Waste Dump, 2019

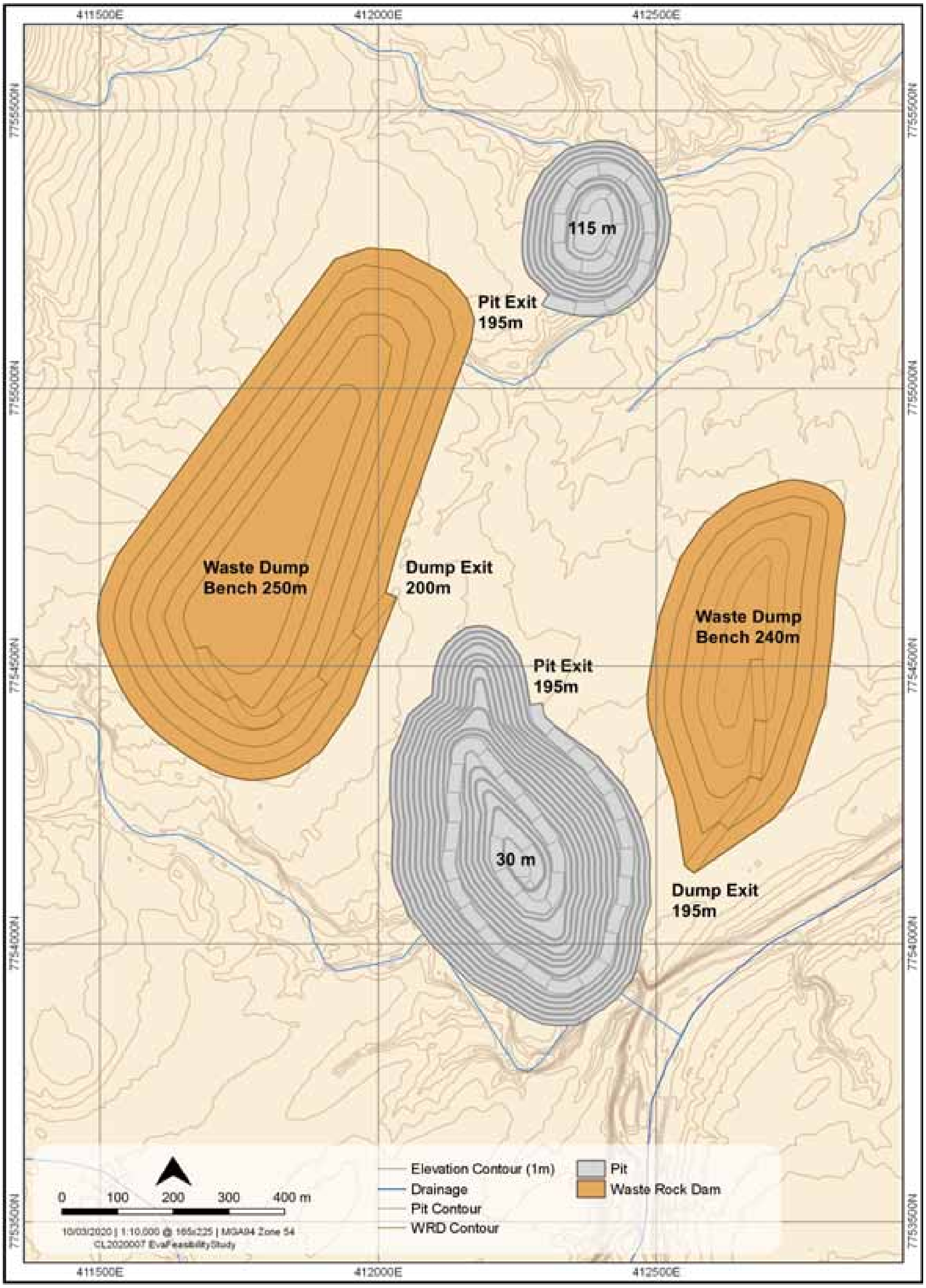


Figure 16-10: Scanlan Pit and Waste Dump, 2019

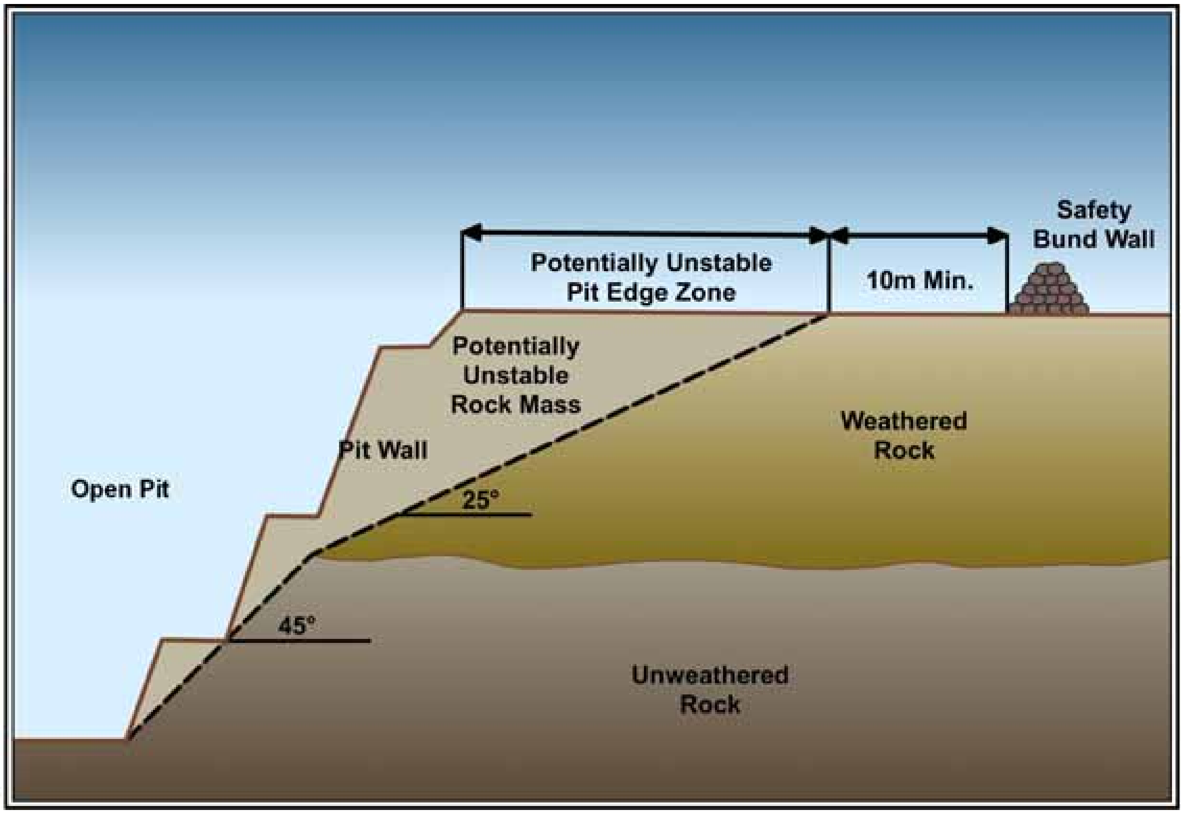


Figure 16-11: Typical Eva Project Pit Configuration in Weathered and Unweathered Rock, 2019

16.5 Pit Dewatering

Pit dewatering at each of the pits will comprised three methods (components). Initially, several dewatering boreholes will be drilled around the pit areas, which will be pumped to facilitate draw-down of the local phreatic surface of the water table. In pit sumps will be the second method of pit dewatering, and the third method will be the installation of horizontal drain holes. These methods will be planned to ensure minimal groundwater reports to the pit walls during operations. This action is a requirement as part of the pit wall structural control.

Mining at the Little Eva and Blackard deposits will necessitate the dewatering of the orebody aquifers. Studies by Morgan (2007 and 2011) have indicated that the principal water-bearing zone is the leached upper contact between the intrusive porphyry body and the overlying calc-silicate rock unit. Pit dewatering will be achieved by sustained pumping rates of 4,000 m^3/d (733 gpm). This will involve both perimeter production bores, in-pit sumps, and sub-horizontal wall drain holes. The wall drain holes would target underdrainage of the broken domain identified in the northern portion of the western pit wall, as well as a fault on the eastern side of the pit that converges with the Coolullah Fault to the north of the pit.

Once the main Little Eva orebody aquifer has been dewatered, groundwater inflow to the pit will markedly reduce, and be confined to the lower parts of pit walls. Reduced groundwater flow conditions will result from the cone of depression extending into the less conductive rocks of the

Corella Formation. Groundwater seepage (inflow) to the pit will be maintained by seasonal recharge from creek flow into the fracture systems below the creek beds bounding the sides of the pit.

In addition to the drawdown boreholes, there will be a need for horizontal drain holes, which will typically be 50 mm or 100 mm in diameter, and may be lined with 25 mm or 50 mm perforated or slotted PVC pipe to maintain open drill holes and free drainage conditions. Sumps will be established at the lowest elevation on the bench and pit bottom, and all water flows from the various aquifer sources will be channeled to the sumps. Depending on the volume of water, these sump pumps will either be submersible, fixed, or mounted on pontoons (or floaters), and they will pump water out of the pit by way of a staged pumping system.

All pits and dumps will have the necessary diversion ditches and bunds, topsoil stockpiles, and sediment control structures.

All mine water will be pumped to settling ponds to be located close to the open pits and will be used for dust suppression for the pits, haul roads, waste dump, and plant. Excess mine water at the Little Eva, Turkey Creek, Bedford, Blackard, and Scanlan pits will be pumped to the stormwater dam for storage and used for process plant requirements. Water from the Lady Clayre and Ivy Ann pits will be used for local dust management, and either stored, made available to the local ranching community, or discharged to the environment (pending receipt of the required environmental approvals).

A detailed pit dewatering plan will be required, which should include in-pit sumps, horizontal drains, and dewatering wells. Capital has been allocated; however, a more formal plan will be developed before development starts. At present, the Eva Copper Project is planning to use surface runoff in the satellite areas (Bedford, Lady Clayre, Ivy Ann, Blackard, and Scanlan) that accumulates in areas such as the in-pit sump in the pit, the sump down gradient of the waste rock pile, and the sedimentation ponds for dust suppression in those areas.

This is also discussed in Section 18.4.2 of this report.

16.6 Life-of-Mine Production Schedule

The mining schedule is based on operating 24 h/d, 365 d/a. The mine and maintenance departments' shift schedules are planned to be two 12-hour shifts per day, with four alternating crews. This schedule is quite common and acceptable in Australia. The Eva Copper Project will provide a safe work environment, and competitive wages and benefits. The Project has used surveys of average salary and wage rates to determine compensations at the mine.

The mining and milling schedules were generated in two different software packages: Datamine's NPVS and Geovia's Mine Scheduler. The mining cost estimator used also verified the computer-generated mining and milling schedules. The primary goals of the mining and milling schedules were the following:

- Maximize the Project's NPV, which involves milling as high a copper grade as feasibly possible. This strategy involves "over mining" to achieve the highest possible mill head grade.
- Maintain as much as possible a blend of 25% native copper ore and 75% sulphide ore; native copper ores are only found in the Blackard and Scanlan deposits. Minimal mining of Blackard and Scanlan is planned, however, the 25% blend of native copper will be achieved.
- Schedule the first two years of production by months, Years three and four by quarters, and the remaining LOM by years.

- Minimize the pre-stripping required.
- Minimize the rehandling required.

The Little Eva pit has six mining phases as a design basis for the LOM production schedule, named Phase 1 through Phase 6. Blackard has three mining phases, Turkey Creek two mining phases, and the other satellite pits are all single-phase mining. Mining will begin in Little Eva, followed by Turkey Creek, Blackard, the Bedford pits, Lady Clayre, and conclude with the mining of Ivy Ann, Scanlan and Blackard. Mining will begin in the Little Eva pit during the pre-production period, and Turkey Creek mining will start in Year 2. Blackard and Scanlan will supply all native copper ores, which are considerably softer (BWI <10 kWh/t) than the ores found in the other deposits. The other satellite pits will be mined on an as-needed basis. All waste haul distances from all pits are short, between 0.5 km and 2.5 km. The Company's mine trucks will haul ore from Little Eva, Turkey Creek, Bedford, Lady Clayre, Blackard, and Scanlan; highway transport trucks will haul ore from Ivy Ann.

There has been no mining at the Eva Copper Project to date. The current estimated mine life is 16 years. The LOM mine schedule strived to maximize the delivered mill head grade and minimize the amount of rehandling material.

The processing operation has been initially scheduled at 10.2 Mt/a (31,000 t/d). The following net smelter return (NSR) cut-off values shown in Table 16-5 were used to generate the LOM schedule.

Table 16-5: Eva NSR Cut-off Values, 2019

Pit	HG Cut-off Value ($/t)	MG Cut-off Value ($/t)	LG Cut-off Value ($/t)
Little Eva (Phases 1-6)	15.50	12.50	8.95
Turkey Creek (Phases 1-2)	15.50	12.50	8.95
Bedford (North/South pits)	15.50	12.50	9.35
Lady Clayre (W, N, S pits)	15.50	12.50	10.32
Ivy Ann	15.50	12.50	11.44
Blackard (Phases 1-3)	15.50	12.50	9.35
Scanlan (North/South pits)	15.50	12.50	10.32

Table 16-6 summarizes the phased design estimate for the LOM production schedule and shows the amount of material and contained copper and gold for each phase, and the periods mined per phase. CMMC believes the LOM production schedule is achievable. Figure 16-12 is the LOM mine schedule.

CMMC generated an independent production schedule (Year 1 to Year 16, with Year -1 and 1 scheduled monthly, Year 2 to Year 4 by quarter, and Year 5 to Year 16 annually) for the Eva Copper Project LOM utilizing Geovia's Miner Scheduler package and a detailed Excel workbook model. It is planned that during basic engineering scheduling-specific software will be utilized to check and refine the LOM schedule. CMMC's phase designs and ultimate pits were used as a guide. Slope stability and pit dewatering will be the major mining risks to achieving the production targets; however, these risks are inherent to most open pit mines, and they are manageable.

Table 16-6: Summary of Phase Designs

Pit	Phase	Ore Tonnes (t '000s)	Cu Grade (%Cu)	Au Grade (g/t)	Waste Tonnes (t '000s)	Total Tonnes (t '000s)	Strip Ratio (W:O)	CM Cu (Mlb)	CM Au (oz '000s)	Years Mined (Year)
Little Eva	1	20,294	0.52	0.08	24,331	44,625	1.2	233	51	0-2
Little Eva	2	26,205	0.38	0.07	41,949	68,153	1.6	220	61	1-5
Little Eva	3	24,116	0.32	0.07	61,951	86,067	2.6	170	50	5-9
Little Eva	4	11,849	0.37	0.07	11,405	23,254	1.0	97	27	9-11
Little Eva	5	2,449	0.31	0.06	5,463	7,912	2.2	17	5	11-12
Little Eva	6	12,805	0.33	0.06	6,346	19,151	0.5	93	23	13
Bedford North	1	2,282	0.58	0.16	8,916	11,198	3.9	29	11	3-5
Bedford South	2	445	0.48	0.15	2,785	3,230	6.3	5	2	5
Lady Clayre	1	3,621	0.45	0.19	14,868	18,489	4.1	36	23	6-8
Ivy Ann	1	2,282	0.43	0.09	6,188	8,471	2.7	22	7	5-6
Turkey Creek	1	5,817	0.48	-	15,701	21,519	2.7	62	0	1-3
Turkey Creek	2	4,879	0.47	-	17,238	22,117	3.5	51	0	3
Blackard	1	18,581	0.57	0	36,435	55,016	2.0	233	0	2-7
Blackard	2	5,623	0.55	0	28,593	34,215	5.1	68	0	7-10
Blackard	3	18,511	0.54	0	62,195	80,706	3.4	220	0	3-14
Scanlan		11,266	0..66	0	39,871	51,137	3.5	164	0	6-13
Total	**11**	**171,025**	**0.46**	**0.08**	**384,235**	**555,260**	**2.2**	**1,718**	**260**	

Note: CM = Contained Metal

It should be noted again that the ultimate pits, the LOM mine production schedule generated by CMMC, and the LOM processing schedule generated by CMMC only use Proven and Probable Mineral Reserves (converted Measured and Indicated Resources). No Inferred Resources and no oxide materials are included in the LOM phase designs, LOM mining schedule, or the LOM processing schedule.

Blending of the native copper and sulphide ores will be handled at the primary crusher pad area. All ore to be mined is expected to be either sulphide (fresh rock), transition, or a native copper zone. All heavily oxidized, weathered material, and most of the transition rock are treated as waste in the LOM plan.

Currently, CMMC is evaluating the economics of the oxidized material within the LOM plan, which is approximately 14.8 Mt grading 0.34% Cu, containing approximately 112 Mlb of copper.

Table 16-7 and Table 16-8 show the 2019 LOM mining and processing schedules for the LOM budget, which are achievable with the projected fleets for the Eva Copper Project. Figure 16-12 is the LOM mine schedule and copper produced.

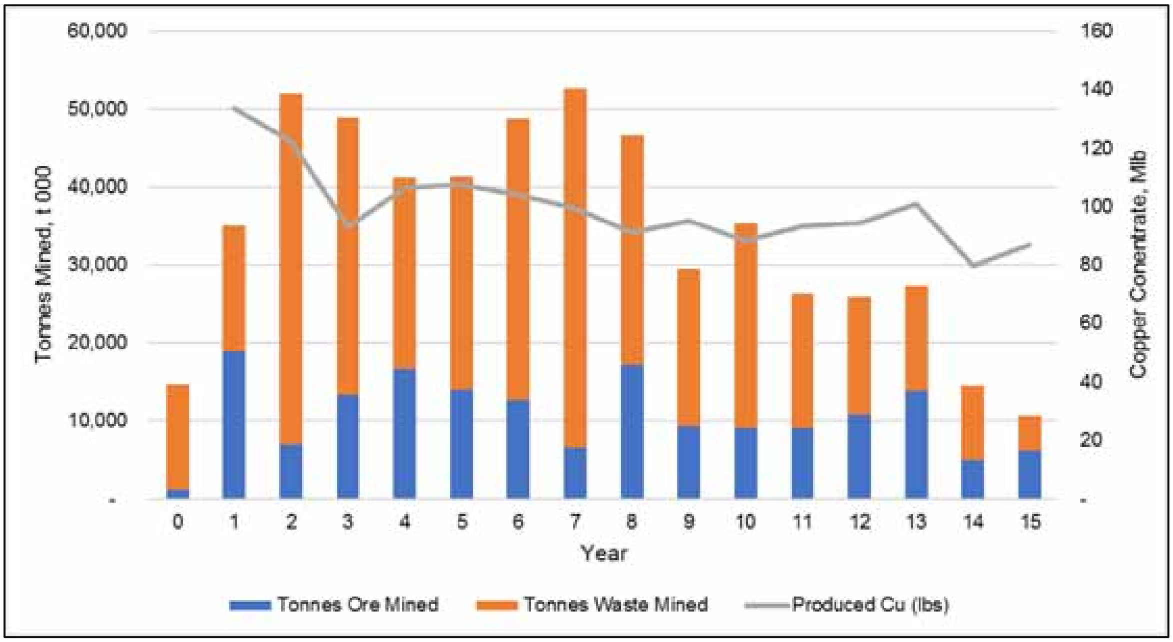


Figure 16-12: Eva Copper Project LOM Mine Schedule and Copper Produced

Table 16-7: LOM Mining Schedule

Deposit	Category	Unit	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Eva Project	Native Tonnes	t '000s	35,560	-	-	1	3,620	3,302	3,011	2,986	798	2,989	2,922	2,790	2,921	2,961	2,975	2,460	1,824
Eva Project	Native Cu Grade	% Cu	0.62	-	-	0.31	0.57	0.61	0.63	0.66	0.52	0.53	0.56	0.61	0.66	0.68	0.63	0.67	0.74
Eva Project	Native Cu Tonnes	t	220,863	-	-	3	20,610	20,302	18,970	19,706	4,174	15,885	16,354	16,892	19,156	20,231	18,711	16,364	13,508
Eva Project	Transition Tonnes	t '000s	2,734	-	-	-	12	45	256	542	36	136	78	61	124	279	491	674	-
Eva Project	Transition Cu Grade	% Cu	0.55	-	-	-	0.47	0.65	0.55	0.51	0.47	0.60	0.86	0.41	0.49	0.55	0.56	0.55	-
Eva Project	Transition Cu Tonnes	t	15,022	-	-	-	58	291	1,408	2,752	168	812	676	251	611	1,548	2,729	3,718	-
Eva Project	Sulphide Tonnes	t '000s	132,091	1,168	18,908	6,898	9,643	13,285	10,700	9,155	5,701	14,172	6,301	6,186	6,020	7,597	10,628	2,058	3,669
Eva Project	Sulphide Cu Grade	% Cu	0.41	0.51	0.53	0.43	0.41	0.41	0.42	0.41	0.35	0.33	0.39	0.36	0.39	0.36	0.38	0.40	0.50
Eva Project	Sulphide Cu Tonnes	t	543,767	5,920	100,981	29,902	39,343	54,488	45,320	37,177	20,062	46,989	24,454	22,165	23,272	26,983	40,232	8,199	18,278
Eva Project	**Total Ore Tonnes**	**t '000s**	**170,386**	**1,168**	**18,908**	**6,899**	**13,275**	**16,632**	**13,966**	**12,683**	**6,535**	**17,296**	**9,301**	**9,038**	**9,066**	**10,838**	**14,095**	**5,192**	**5,494**
Eva Project	**Total Ore Cu Grade**	**% Cu**	**0.46**	**0.51**	**0.53**	**0.43**	**0.45**	**0.45**	**0.47**	**0.47**	**0.37**	**0.37**	**0.45**	**0.43**	**0.47**	**0.45**	**0.44**	**0.54**	**0.58**
Eva Project	**Total Ore Cu Tonnes**	**t**	**779,653**	**5,920**	**100,981**	**29,904**	**60,010**	**75,081**	**65,699**	**59,636**	**24,404**	**63,686**	**41,484**	**39,308**	**43,038**	**48,762**	**61,672**	**28,281**	**31,786**
Eva Project	Waste Tonnes	t '000s	380,574	13,520	16,132	45,113	35,669	24,541	27,339	36,100	46,185	29,424	20,233	26,265	17,148	15,077	13,245	9,408	5,174
Eva Project	**Total Tonnes**	**t '000s**	**550,959**	**14,688**	**35,040**	**52,012**	**48,943**	**41,174**	**41,228**	**46,671**	**52,720**	**46,720**	**29,534**	**35,303**	**26,214**	**25,915**	**27,340**	**14,600**	**10,668**
Eva Project	Sulphide Au Grade	g/t	0.05	0.07	0.08	0.02	0.02	0.06	0.06	0.03	0.08	0.07	0.03	0.04	0.05	0.04	0.04	-	-
Eva Project	Sulphide Au Grams	g	8,083,938	83,892	1,523,098	143,398	293,169	1,071,467	820,904	387,155	511,110	1,275,763	279,997	325,854	414,360	395,217	558,553	-	-
Eva Project	Sulphide Au Ounces	oz '000s	260	3	49	5	9	34	26	12	16	41	9	10	13	13	18	-	-
Blackard	Native Tonnes	t '000s	25,188			0.82	3,620	3,302	3,011	2,982	256	1,799	1,686	901	966	1,127	1,253	2,460	1,824
Blackard	Native Cu Grade	% Cu	0.61			0.31	0.57	0.61	0.63	0.66	0.44	0.52	0.54	0.51	0.53	0.56	0.60	0.67	0.74
Blackard	Native Cu Tonnes	t	152,443	-	-	3	20,610	20,302	18,970	19,688	1,135	9,292	9,077	4,632	5,121	6,285	7,459	16,364	13,508
Blackard	Transition Tonnes	t '000s	2,474				12	45	256	542	36	136	78	59	77	174	385	674	-
Blackard	Transition Cu Grade	% Cu	0.53				0.47	0.65	0.55	0.51	0.47	0.60	0.86	0.40	0.40	0.42	0.50	0.55	-
Blackard	Transition Cu Tonnes	t	13,105	-	-	-	58	291	1,408	2,752	168	812	676	239	305	735	1,943	3,718	-
Blackard	Sulphide Tonnes	t '000s	14,860				96	120	385	4,177	38	99	1,262	843	608	618	885	2,058	3,669
Blackard	Sulphide Cu Grade	% Cu	0.48				0.37	0.56	0.51	0.45	0.41	0.36	0.58	0.65	0.53	0.45	0.40	0.40	0.50
Blackard	Sulphide Cu Tonnes	t	70,937	-	-	-	356	668	1,956	18,694	157	355	7,275	5,474	3,211	2,797	3,517	8,199	18,278
Blackard	**Total Ore Tonnes**	**t '000s**	**42,522**	**-**	**-**	**1**	**3,729**	**3,468**	**3,651**	**7,700**	**330**	**2,034**	**3,026**	**1,804**	**1,651**	**1,920**	**2,523**	**5,192**	**5,494**

Deposit	Category	Unit	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Blackard	**Total Ore Cu Grade**	**% Cu**	**0.56**	**-**	**-**	**0.31**	**0.56**	**0.61**	**0.61**	**0.53**	**0.44**	**0.51**	**0.56**	**0.57**	**0.52**	**0.51**	**0.51**	**0.54**	**0.58**
Blackard	**Total Ore Cu Tonnes**	**t**	**236,485**	**-**	**-**	**3**	**21,024**	**21,261**	**22,334**	**41,134**	**1,460**	**10,459**	**17,028**	**10,344**	**8,636**	**9,817**	**12,918**	**28,281**	**31,786**
Blackard	Waste Tonnes	t '000s	131,256		-	16,972	10,079	2,666	2,614	6,042	17,255	9,646	11,391	15,978	12,038	6,475	5,518	9,408	5,174
Blackard	**Total Tonnes**	**t '000s**	**173,778**			**16,972**	**13,807**	**6,134**	**6,265**	**13,743**	**17,584**	**11,680**	**14,417**	**17,783**	**13,688**	**8,395**	**8,040**	**14,600**	**10,668**
Scanlan	Native Tonnes	t '000s	10,372							4	542	1,190	1,236	1,889	1,955	1,834	1,722		
Scanlan	Native Cu Grade	% Cu	0.66							0.49	0.56	0.55	0.59	0.65	0.72	0.76	0.65		
Scanlan	Native Cu Tonnes	t	68,421	-	-	-	-	-	-	19	3,039	6,593	7,276	12,260	14,035	13,947	11,252	-	-
Scanlan	Transition Tonnes	t '000s	260											2	47	105	106		
Scanlan	Transition Cu Grade	% Cu	0.74											0.69	0.64	0.77	0.74		
Scanlan	Transition Cu Tonnes	t	1,918	-	-	-	-	-	-	-	-	-	-	12	306	813	786	-	-
Scanlan	Sulphide Tonnes	t '000s	639											0.31	23	91	524	-	-
Scanlan	Sulphide Cu Grade	% Cu	0.63											0.28	0.31	0.47	0.67	-	-
Scanlan	Sulphide Cu Tonnes	t	4,010	-	-	-	-	-	-	-	-	-	-	1	73	425	3,511	-	-
Scanlan	**Total Ore Tonnes**	**t '000s**	**11,271**	**-**	**-**	**-**	**-**	**-**	**-**	**4**	**542**	**1,190**	**1,236**	**1,891**	**2,026**	**2,030**	**2,353**	**-**	**-**
Scanlan	**Total Ore Cu Grade**	**% Cu**	**0.66**	**-**	**-**	**-**	**-**	**-**	**-**	**0.49**	**0.56**	**0.55**	**0.59**	**0.65**	**0.71**	**0.75**	**0.66**	**-**	**-**
Scanlan	**Total Ore Cu Tonnes**	**t**	**74,348**	**-**	**-**	**-**	**-**	**-**	**-**	**19**	**3,039**	**6,593**	**7,276**	**12,273**	**14,414**	**15,185**	**15,549**	**-**	**-**
Scanlan	Waste Tonnes	t '000s	30,352	-	-	-	-	-	-	3,911	8,242	3,617	2,342	2,983	2,263	1,474	5,520	-	-
Scanlan	**Total Tonnes**	**t '000s**	**41,623**							**3,915**	**8,784**	**4,806**	**3,577**	**4,874**	**4,290**	**3,504**	**7,873**	**-**	**-**
Turkey Creek	Native Tonnes	t '000s	-																
Turkey Creek	Native Cu Grade	% Cu	-																
Turkey Creek	Native Cu Tonnes	t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Turkey Creek	Transition Tonnes	t '000s	-																
Turkey Creek	Transition Cu Grade	% Cu	-																
Turkey Creek	Transition Cu Tonnes	t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Turkey Creek	Sulphide Tonnes	t '000s	10,688			4,997	5,691												
Turkey Creek	Sulphide Cu Grade	% Cu	0.48			0.48	0.47												
Turkey Creek	Sulphide Cu Tonnes	t	50,853	-	-	24,212	26,641	-	-	-	-	-	-	-	-	-	-	-	-
Turkey Creek	**Total Ore Tonnes**	**t '000s**	**10,688**	**-**	**-**	**4,997**	**5,691**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Turkey Creek	**Total Ore Cu Grade**	**% Cu**	**0.48**	**-**	**-**	**0.48**	**0.47**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**

Deposit	Category	Unit	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Turkey Creek	**Total Ore Cu Tonnes**	**t**	**50,853**	**-**	**-**	**24,212**	**26,641**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Turkey Creek	Waste Tonnes	t '000s	33,021	-	2,865	12,523	17,633	-	-	-	-	-	-	-	-	-	-	-	
Turkey Creek	**Total Tonnes**	**t '000s**	**43,709**		**2,865**	**17,520**	**23,324**												
Little Eva	Native Tonnes	t '000s	-																
Little Eva	Native Cu Grade	% Cu	-																
Little Eva	Native Cu Tonnes	t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-		
Little Eva	Transition Tonnes	t '000s	-																
Little Eva	Transition Cu Grade	% Cu	-																
Little Eva	Transition Cu Tonnes	t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-		
Little Eva	Sulphide Tonnes	t '000s	97,562	1,168	18,908	1,902	3,855	11,623	9,121	2,840	4,298	11,969	5,039	5,342	5,389	6,888	9,219		
Little Eva	Sulphide Cu Grade	% Cu	0.39	0.51	0.53	0.30	0.32	0.38	0.41	0.32	0.32	0.30	0.34	0.31	0.37	0.34	0.36		
Little Eva	Sulphide Cu Tonnes	t	377,010	5,920	100,981	5,690	12,345	44,120	37,585	9,140	13,946	36,460	17,180	16,691	19,988	23,760	33,205		
Little Eva	**Total Ore Tonnes**	**t '000s**	**97,562**	**1,168**	**18,908**	**1,902**	**3,855**	**11,623**	**9,121**	**2,840**	**4,298**	**11,969**	**5,039**	**5,342**	**5,389**	**6,888**	**9,219**		
Little Eva	**Total Ore Cu Grade**	**% Cu**	**0.39**	**0.51**	**0.53**	**0.30**	**0.32**	**0.38**	**0.41**	**0.32**	**0.32**	**0.30**	**0.34**	**0.31**	**0.37**	**0.34**	**0.36**		
Little Eva	**Total Ore Cu Tonnes**	**t**	**377,010**	**5,920**	**100,981**	**5,690**	**12,345**	**44,120**	**37,585**	**9,140**	**13,946**	**36,460**	**17,180**	**16,691**	**19,988**	**23,760**	**33,205**	**-**	**-**
Little Eva	Waste Tonnes	t '000s	152,662	13,520	13,267	15,618	7,147	14,657	17,159	19,524	13,270	12,514	6,500	7,304	2,847	7,128	2,208		
Little Eva	**Total Tonnes**	**t '000s**	**250,224**	**14,688**	**32,175**	**17,520**	**11,002**	**26,280**	**26,280**	**22,365**	**17,568**	**24,483**	**11,539**	**12,646**	**8,236**	**14,016**	**11,427**		
Little Eva	Sulphide Au Grade	g/t	0.07	0.07	0.08	0.08	0.08	0.07	0.07	0.07	0.07	0.07	0.06	0.06	0.08	0.06	0.06		
Little Eva	Sulphide Au Grams	g	6,781,557	83,892	1,523,098	143,398	293,169	827,000	655,961	187,454	290,093	803,511	279,997	325,854	414,360	395,217	558,553	-	-
Little Eva	Sulphide Au Ounces	oz '000s	218	2.7	49.0	4.6	9.4	26.6	21.1	6.0	9.3	25.8	9.0	10.5	13.3	12.7	18.0	-	-
Bedford	Native Tonnes	t '000s	-																
Bedford	Native Cu Grade	% Cu	-																
Bedford	Native Cu Tonnes	t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Bedford	Transition Tonnes	t '000s	-																
Bedford	Transition Cu Grade	% Cu	-																
Bedford	Transition Cu Tonnes	t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Bedford	Sulphide Tonnes	t '000s	2,659				-	1,542	1,117										
Bedford	Sulphide Cu Grade	% Cu	0.57				-	0.63	0.49										
Bedford	Sulphide Cu Tonnes	t	15,173	-	-	-	-	9,700	5,473	-	-	-	-	-	-	-	-	-	-

Deposit	Category	Unit	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Bedford	**Total Ore Tonnes**	**t '000s**	**2,659**	-	-	-	-	**1,542**	**1,117**	-	-	-	-	-	-	-	-	-	-
Bedford	**Total Ore Cu Grade**	**% Cu**	**0.57**	-	-	-	-	**0.63**	**0.49**	-	-	-	-	-	-	-	-	-	-
Bedford	**Total Ore Cu Tonnes**	**t**	**15,173**	-	-	-	-	**9,700**	**5,473**	-	-	-	-	-	-	-	-	-	-
Bedford	Waste Tonnes	t '000s	11,798				810	7,218	3,769										
Bedford	**Total Tonnes**	**t '000s**	**14,456**				**810**	**8,760**	**4,886**										
Bedford	Sulphide Au Grade	g/t	0.15				-	0.16	0.14										
Bedford	Sulphide Au Grams	g	402,782	-	-	-	-	244,468	158,315	-	-	-	-	-	-	-	-	-	-
Bedford	Sulphide Au Ounces	t '000s	13	-	-	-	-	7.9	5.1	-	-	-	-	-	-	-	-	-	-
Lady Clayre	Native Tonnes	t '000s	-																
Lady Clayre	Native Cu Grade	% Cu	-																
Lady Clayre	Native Cu Tonnes	t	-																
Lady Clayre	Transition Tonnes	t '000s	-																
Lady Clayre	Transition Cu Grade	% Cu	-																
Lady Clayre	Transition Cu Tonnes	t	-																
Lady Clayre	Sulphide Tonnes	t '000s	3,495							26	1,365	2,104							
Lady Clayre	Sulphide Cu Grade	% Cu	0.46							0.35	0.44	0.48							
Lady Clayre	Sulphide Cu Tonnes	t	16,226	-	-	-	-	-	-	92	5,959	10,175	-	-	-	-	-	-	-
Lady Clayre	**Total Ore Tonnes**	**t '000s**	**3,495**	-	-	-	-	-	-	**26**	**1,365**	**2,104**	-	-	-	-	-	-	-
Lady Clayre	**Total Ore Cu Grade**	**% Cu**	**0.46**	-	-	-	-	-	-	**0.35**	**0.44**	**0.48**	-	-	-	-	-	-	-
Lady Clayre	**Total Ore Cu Tonnes**	**t**	**16,226**	-	-	-	-	-	-	**92**	**5,959**	**10,175**	-	-	-	-	-	-	-
Lady Clayre	Waste Tonnes	t '000s	14,955	-	-	-	-	-	-	3,889	7,419	3,647	-	-	-	-	-	-	-
Lady Clayre	**Total Tonnes**	**t '000s**	**18,450**							**3,915**	**8,784**	**5,751**							
Lady Clayre	Sulphide Au Grade	g/t	0.20							0.05	0.16	0.22							
Lady Clayre	Sulphide Au Grams	g	694,625	-	-	-	-	-	-	1,355	221,018	472,252	-	-	-	-	-	-	-
Lady Clayre	Sulphide Au Ounces	oz '000s	22	-	-	-	-	-	-	0.0	7.1	15.2	-	-	-	-	-	-	-

Deposit	Category	Unit	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Ivy Ann	Native Tonnes	t '000s	-																
Ivy Ann	Native Cu Grade	% Cu	-																
Ivy Ann	Native Cu Tonnes	t	-																
Ivy Ann	Transition Tonnes	t '000s	-																
Ivy Ann	Transition Cu Grade	% Cu	-																
Ivy Ann	Transition Cu Tonnes	t	-																
Ivy Ann	Sulphide Tonnes	t '000s	2,189						77	2,112									
Ivy Ann	Sulphide Cu Grade	% Cu	0.44						0.40	0.44									
Ivy Ann	Sulphide Cu Tonnes	t	9,557	-	-	-	-	-	306	9,250	-	-	-	-	-	-	-	-	-
Ivy Ann	**Total Ore Tonnes**	**t '000s**	**2,189**	**-**	**-**	**-**	**-**	**-**	**77**	**2,112**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Ivy Ann	**Total Ore Cu Grade**	**% Cu**	**0.44**	**-**	**-**	**-**	**-**	**-**	**0.40**	**0.44**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Ivy Ann	**Total Ore Cu Tonnes**	**t**	**9,557**	**-**	**-**	**-**	**-**	**-**	**306**	**9,250**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Ivy Ann	Waste Tonnes	t '000s	6,530						3,797	2,733									
Ivy Ann	**Total Tonnes**	**t '000s**	**8,718**						**3,874**	**4,845**									
Ivy Ann	Sulphide Au Grade	g/t	0.09						0.09	0.09									
Ivy Ann	Sulphide Au Grams	g	204,974	-	-	-	-	-	6,628	198,346	-	-	-	-	-	-	-	-	-
Ivy Ann	Sulphide Au Ounces	oz '000s	7	-	-	-	-	-	0.2	6.4	-	-	-	-	-	-	-	-	-

Notes: **1.** Includes oxidized, some transition, low-grade mineralization, and Inferred Mineral Resources in the waste tonnes.
2. Proven and Probable Mineral Reserves are included as ore at the following NSR cut-off values: $8.95/t for Little Eva and Turkey Creek; $9.35/t for Bedford and Blackard; $10.32/t for Lady Clayre and Scanlan; and $11.44/t for Ivy Ann.
3. Numbers may not add due to rounding.

Table 16-8: LOM Processing Schedule, 2019

	Unit	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Tonnes Ore Mined	kt	**170,386**	1,168	18,908	6,899	13,275	16,632	13,966	12,683	6,499	17,196	9,359	9,057	9,048	10,740	13,884	4,903	6,168
Tonnes Waste Mined	kt	**380,574**	13,520	16,132	45,113	35,669	24,541	27,339	36,100	46,221	29,524	20,175	26,246	17,166	15,175	13,456	9,697	4,501
Total Material Mined	kt	**550,959**	14,688	35,040	52,012	48,943	41,174	41,305	48,783	52,720	46,720	29,534	35,303	26,214	25,915	27,340	14,600	10,668
Stripping Ratio	(W:O)	**2.2**	11.6	0.9	6.5	2.7	1.5	2.0	2.8	7.1	1.7	2.2	2.9	1.9	1.4	1.0	2.0	0.7
Tonnes Moved per Day	t/d	**96,845**	79,826	96,000	142,500	133,725	112,805	113,165	133,651	144,044	128,000	80,915	96,720	71,622	71,000	74,903	40,000	30,649
Milling and Production																		
Dry Tonnes Milled	kt	**170,386**	-	11,388	11,388	11,419	11,388	11,388	11,388	11,419	11,388	11,388	11,388	11,419	11,388	11,388	11,388	10,860
Re-handle Tonnes	kt	**31,833**	-	1,726	4,843	1,788	-	-	-	4,920	-	2,029	2,331	2,371	648	-	6,485	4,692
Percent Re-handle	%	**18**	0	15	43	16	0	0	0	43	0	18	20%	21	6	0	57	43
Native Copper Tonnes	kt	**35,560**	-	-	1	2,833	2,896	2,800	2,986	2,201	2,989	2,922	2,790	2,921	2,961	2,975	2,460	1,824
Native Copper Percent	%		0	0	0	25	25	25	26	19	26	26	25	26	26	26	22	17
Tonnes Milled per Day	t/d		-	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200	31,200
Head Grades																		
Head Grade - Cu (%)	Cu%	**0.46**	-	0.56	0.53	0.44	0.49	0.50	0.49	0.45	0.42	0.44	0.41	0.43	0.44	0.47	0.37	0.42
Head Grade - Au (g/t)	Au g/t	**0.047**	-	0.083	0.052	0.028	0.066	0.059	0.028	0.067	0.079	0.039	0.040	0.047	0.037	0.038	0.027	0.019
Model Cu Recovery (POC) - (%)	Avg. Cu Recov. %	**87.1**	-	95	92	85	87	87	84	88	86	86	87	86	86	86	86	87
Head Grade - Density	t/m^3	**2.6**	-	2.8	2.7	2.6	2.6	2.6	2.5	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6
Contained Copper	Mlb	**1,715**		140.50	133.17	109.91	122.74	124.33	124.18	112.95	105.87	111.01	101.99	108.19	109.61	118.01	92.47	100.52
Recoveries																		
Recovery - Cu (%)	%	**87**		95	92	85	87	87	84	88	86	86	87	86	86	86	86	87
Recovery - Au (%)	%	**78**		78	78	78	78	78	78	78	78	78	78	78	78	78	78	78
Produced Metal																		
Produced Cu (lb)	Mlb	**1,497**		133.48	122.46	93.02	106.48	107.60	104.08	99.35	91.20	95.30	88.25	93.31	94.30	100.93	79.88	87.03
Produced Au (oz)	koz	**203**		23.63	14.72	8.05	18.94	16.77	7.93	19.32	22.68	11.08	11.46	13.58	10.64	10.79	7.77	5.18
Concentrate Produced																		
Concentrate Produced (DMT)	DMT 000	**2,425**		216.2	198.4	150.7	172.5	174.3	168.6	160.9	147.7	154.4	143.0	151.2	152.8	163.5	129.4	141.0
Concentrate Produced (WMT)	WMT 000	**2,650**		236.3	216.8	164.7	188.5	190.5	184.3	175.9	161.5	168.7	156.2	165.2	167.0	178.7	141.4	154.1
Concentrate Grade	%	**28**		28	28	28	28	28	28	28	28	28	28	28	28	28	28	28
Moisture %	%	**8.5**		8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.5
Payable Metal																		
Payable - Cu (lb)	Mlb	**1,437**		128.14	117.56	89.30	102.22	103.30	99.92	95.37	87.55	91.49	84.72	89.58	90.53	96.89	76.69	83.55
Payable - Au (oz)	koz	**192**		22.45	13.99	7.65	18.00	15.94	7.53	18.35	21.54	10.52	10.88	12.90	10.11	10.25	7.38	4.92

Notes: **1.** Milled tonnes do not include oxidized, low-grade mineralization, or Inferred Mineral Resources. **2.** Contains stockpile and re-handle material. **3.** Copper recoveries of 95% for sulphide and 63% for native copper materials.
Production: M = months; Q = quarters; Y = years

16.7 Mine Equipment

Conventional open pit mining methods will be employed at the Eva Copper Project's open pits. The method of extraction will consist of conventional blasthole drilling, blasting, shovel loading, and off-highway rear-dump hauling. As an example, the current fleet may be composed of Komatsu PC4000 (22 m^3) hydraulic shovels, Komatsu HD1500 (141 tonne) mechanical rear-dump trucks, and Atlas Copco PV-271 (250 mm/9.75-inch diameter) diesel drills to drill, blast, load, and haul ore and waste material to designated locations.

Diesel-powered equipment was selected at this time due to the high electrical power rates in Queensland. The prices of the individual pieces of equipment was also a factor in selecting the fleets. The ability to selectively mine the IOCG (iron-oxide-copper-gold) deposits was also considered. As the Project progresses into detailed engineering, further refinement of equipment selection will be performed.

CMMC believes that the mining method and mine design criteria, including bench heights, road widths, and pit slopes, are appropriate for mining of the orebody. All major equipment will be purchased new; however, CMMC may purchase some equipment used, if the condition and price are favourable. Table 16-9 shows a list of the major mining equipment for the Eva Copper Project in Year 2, which is very close to full capacity.

Table 16-9: Major Mining Equipment Estimate (Year 2), January 2020

No. of Units	Type	Make/Model (Example Only)	Descriptions
3	Front Shovel	Komatsu PC4000-	22-m^3
2	Front End Loader	Komatsu WA900	13-m^3
1	Front End Loader	Komatsu WA600	6.5-m^3
19	Truck	Komatsu HD1500-7	141-t
2	Water Truck	Komatsu HD605	63-t
4	Track Dozer	Komatsu D375-A	605 HP, 22-m^3
1	Wheel Loader	Komatsu WA600-5	530 HP
3	Motor Grader	Komatsu 825A-2	209 HP, 2.4-m
3	Blasthole Drill	Atlas Copco PV271	75 klb pulldown
1	Blasthole Drill (Small Patterns, Trim, Presplit)	Atlas Copco DM45	45 klb pulldown
1	Stemming Loader	Komatsu WA200PZ	
1	Fuel/Lube Truck	Komatsu HD605-7E	
1	Small Excavator	Komatsu PC850-8	
4	ADT Wiggle Wagon	Komatsu HM400-3MO	40-t
1	Tire Handler	Komatsu WA600	63-t
1	Portable Crush/Screen Plant		
1	Compactor	Komatsu WF450-T	
1	Mechanic's Truck		
2	Shovel Crew Hiab		
1	Software, Light Vehicles, etc.		

Note: Additional support and ancillary equipment will be required.

The LOM estimated 22 m^3 excavator productivity is 2,417 tonnes per operating hour (t/h), which appears reasonable to CMMC. The LOM truck productivity used is 319 t/h. It is estimated that on average 11.4 Mt of ore and 24.5 Mt of waste will be mined per year. The Project plans to mine a total of 98,000 t/d of ore and waste at a stripping ratio of 2.2:1 w:o. Mineralized waste and low-grade material will be stockpiled for possible processing at the end of the mine life, provided copper and gold prices increase. LOM haul truck availabilities and utilizations are acceptable to CMMC at approximately 92% and 90%, respectively. LOM hydraulic front shovel availabilities and utilizations are acceptable to CMMC at approximately 85% and 90%, respectively.

Blasting services will be contracted.

16.8 Mining Labour Force

Table 16-10 shows the mine labour force in Year 2, a total of approximately 240 employees.

Table 16-10: Eva Mine Labour Summary, Year 2 – January 2020

Area	Hourly	Salaried	Total
Mine Operations	154	-	154
Mine Maintenance	49	-	49
Support	6	31	37
Total	**209**	**31**	**240**

17 RECOVERY METHODS

17.1 Introduction

The Eva Copper Project, comprising Little Eva, Blackard, and other satellite deposits, has been developed based on the mine plan for a nominal combined mining rate of approximately 31,200 t/d copper ore, equivalent to 11.4 Mt/a, with direct feeding of the ore to the processing plant. The processing plant was designed to produce a marketable concentrate with a grade of 28% Cu (and 3 g/t Au when treating gold-bearing ores) using conventional recovery methods, including crushing, grinding, gravity concentration, flotation, and tailings disposal.

The throughput of 31,200 t/d of copper ore was developed for a feed blend consisting of 75% sulphide ores and 25% native copper ores. Throughput was modelled using Ausenco's in-house comminution program, Ausgrind, which was based on breakage testwork obtained from drill hole data compiled from 44 sample sets distributed across the pit.

The design of the processing plant is specific to the treatment of the blended ore, and is based on the results of testwork undertaken to date on sample material originating from the various deposits of sulphide and native copper ores.

17.2 Summary

The unit processes selected were based on the results of metallurgical testwork programs completed between 1996 and 2016, and the results obtained from the Copper Mountain Mine testwork conducted between 2018 and 2019. The treatment plant will consist of three-stage crushing, including High Pressure Grinding Rolls (HPGR), grinding using a ball mill circuit, incorporating a jigging circuit to recover native copper concentrate, followed by flotation and regrind circuits to recover the sulphide copper minerals to a saleable copper concentrate. A gravity concentrator will be included in the regrind circuit to recover any native copper that reports to the flotation circuit. The gravity concentrate will be dewatered and air dried in a paddock prior to shipping. Flotation concentrate will be thickened, filtered, and stockpiled prior to shipping to market. Flotation tailings will be stored as thickened slurry in a tailings storage facility (TSF).

The process facility will recycle water and process solutions as much as possible to minimize fresh water input. Process water will be recycled from the tailings thickener overflow, and supplemented with process water recovered from the TSF. Overflow solution from the concentrate thickener will be recycled directly for use in the grinding circuit. Make-up water, when required, will be provided from water borefields at site. Fresh water will be used for gland service to the slurry pumps, reagent preparation, lube system cooling, filter press cloth wash, and process water make-up.

The process plant will consist of the following unit operations and facilities:

- Gyratory primary crusher with a rock-breaker unit
- Secondary crushing and screening in closed circuit
- Crushed ore stockpile and reclaim
- HPGR circuit in a closed circuit with wet screens
- Ball milling circuit in a closed circuit with cyclones

- A three-stage jigging circuit incorporated into the ball milling circuit
- Rougher flotation
- Concentrate regrind
- Cleaner, recleaner, and cleaner-scavenger Direct Flotation Reactor (DFR) circuit
- Gravity concentration (Knelson) incorporated into the concentrate regrind circuit
- Flotation concentrate thickening and filtration
- Gravity concentrate dewatering
- Concentrate load-out and storage
- Gravity and flotation concentrates dispatch
- Tailings thickening and storage at the TSF
- Reagents make-up, storage, and distribution
- Grinding media storage and addition
- Water services
- Air services (including compressed air and low-pressure process air)
- Water storage facilities
- Tailings storage facility (TSF)
- TSF water reclaim system.

The simplified process flowsheet is shown in Figure 17-1 and a 3D layout view of the processing plant is shown in Figure 17-2.

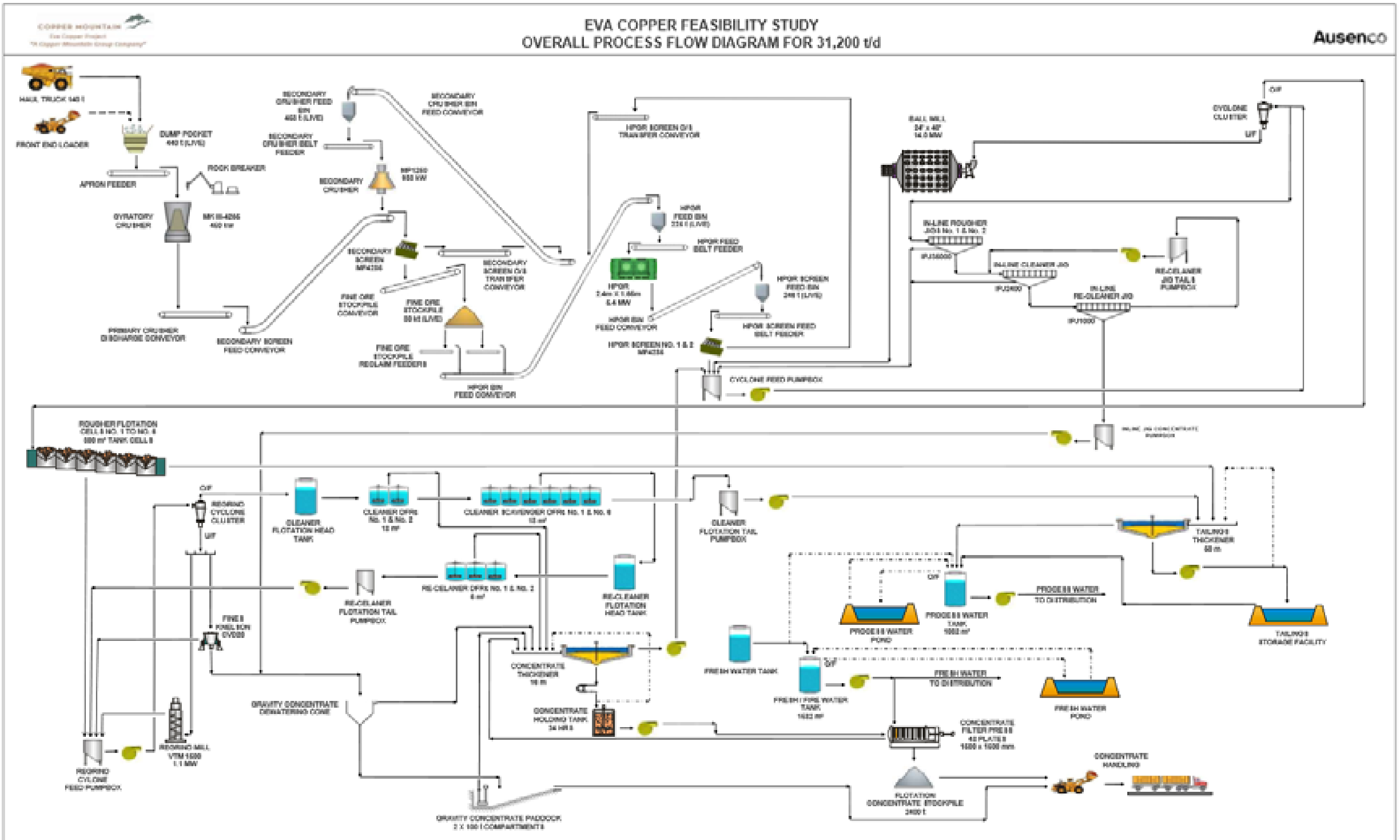


Figure 17-1: Process Flow Diagram

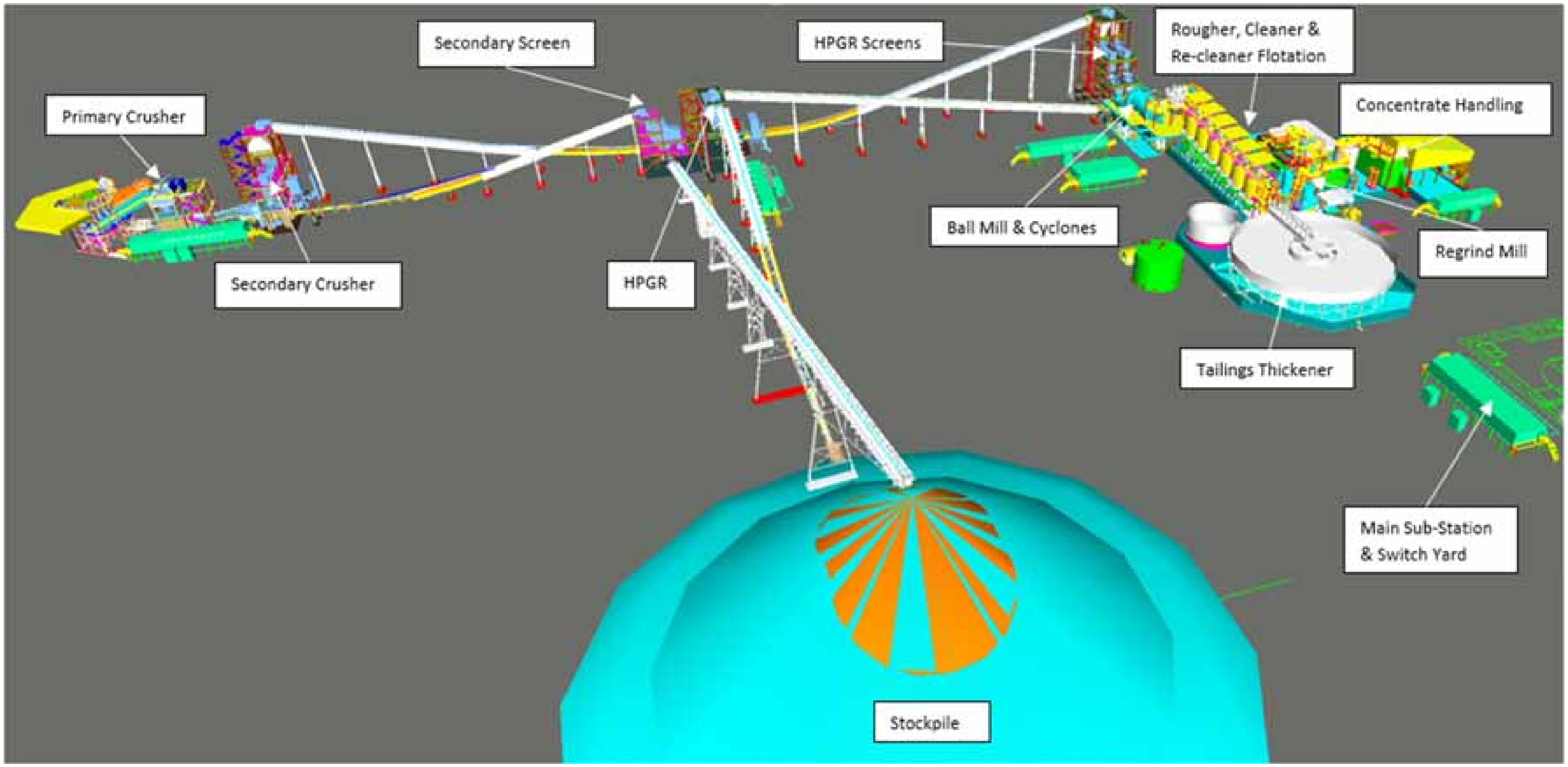


Figure 17-2: Process Plant 3D Layout

The major design criteria are based on those determined by Hatch in the 2018 Feasibility Study, but with incorporation of recent testwork results to accommodate the change in feed materials to include native copper-bearing ores, and the incorporation of HPGR equipment into the flowsheet. Hatch had assessed the testwork results and initially selected suitable process design criteria inputs, which formed the basis for their process flowsheet. Ausenco then reviewed the design data and revised the criteria as necessary considering new testwork results and the feed blend comprising sulphide and native copper ores. Table 17-1, Table 17-2, and Table 17-3 provide a summary of the plant and equipment design criteria for a throughput rate of 31,200 t/d (11.4 Mt/a).

Table 17-1: Physical Properties

Parameter	Application	Unit	Value
Ore Bulk Density	Volumetric design	t/m^3	1.6
	Mass design	t/m^3	1.9
Drawdown Angle	Crushed ore	degrees	60
Angle of Repose	Crushed ore	degrees	38
Drop Weight Index	DWi	kWh/m^3	5.8
Bond Work Indices	Crushing	kWh/t	14.0
	Rod mill	kWh/t	18.7
	Ball mill	kWh/t	16.5
Specific Throughput (M-dot)	HPGR	ts/m^{3}h	291
Bond Abrasion Index	Ai	g	0.17
Gravity Concentrate Density	Unconsolidated	t/m^3	3.00
	Consolidated	t/m^3	4.00
Flotation Concentrate Density	Unconsolidated	t/m^3	1.90
	Consolidated	t/m^3	2.30

Table 17-2: Throughput Rates, Crushing, HPGR, and Grinding Stages

Parameter	Unit	Value
Annual throughput	Mt/a	11.4
Daily throughput	t/d	31,200
Primary and Secondary Crushing Feed		
Availability	%	75
Hourly throughput	t/h	1,733
HPGR and Grinding Feed		
Availability	%	92
Hourly throughput	t/h	1,413

Table 17-3: Production Design Parameters

Parameter	Unit	Value
Feed Grade		
Copper	%	0.56 nominal 0.46 average
Gold	g/t	0.077 nominal 0.047 average
Primary Crushing		
Feed Size (F_{100})	mm	800
Feed Size (F_{80})	mm	423
Product Size (P_{100})	mm	383
Product Size (P_{80})	mm	137
Secondary Crushing		
Product Size (P_{100})	mm	50
Product Size (P_{80})	mm	35
HPGR		
Product Size (P_{100})	mm	6
Product Size (P_{80})	mm	4
Ball Milling		
Product Size (P_{80})	µm	165
Tower Milling (Regrind)		
Product Size (P_{80})	µm	53
Concentrate Production		
Gravity Concentrate	dmt/a	1,484
Flotation Concentrate	dmt/a	192,129
Recovery to Final Concentrate[1]		
Copper	%	87.0
Gold	%	78.0
Final Concentrate Grade[1]		
Copper	%	28
Gold	g/t	3
Final Concentrate Moisture		
% water	w/w	9
Thickener Underflow Density		
% solids	w/w	63

Note: [1]Combined gravity and flotation concentrate grade.

The main design parameters that have changed from the 2018 Feasibility Study are the following:

- The processing plant throughput was increased from 28,000 t/d to 31,200 t/d.

- The comminution flowsheet was changed from a SAG-ball mill grinding circuit with pebble crushing to a three-stage crushing circuit incorporating HPGR followed by ball milling.
- The ball mill dimensions have been increased from 3.7 m diameter by 7.3 m effective grinding length (EGL) to 7.3 m diameter by 12.2 m EGL to handle the product from the HPGR circuit.
- The ball milling product size has been decreased to 80% passing (P_{80}) 165 μm.
- Primary and secondary crushing are closed circuits with availability of 75%, which will require re-handle at the ROM pad by front-end loaders (FELs).
- A dust collection system will be installed at various points of the crushing circuit.
- The coarse ore stockpile has been renamed as fine ore.
- Flash flotation has been removed from the process plant design.
- A jigging circuit was included to process a portion of the cyclone cluster feed, which is limited to 500 t/h due to the maximum capacity of the rougher jigs.
- The rougher flotation retention time has been increased, providing additional capacity to float potential oxides with NaHS.
- The cleaner flotation column and cleaner-scavenger flotation cells have been replaced with a cleaner, recleaner, and cleaner-scavenger DFR circuit to produce a final flotation concentrate.
- A fines gravity concentrator was included in the regrind circuit to process a portion of the cyclone underflow in order to recover any native copper that reported to the flotation rougher concentrate.
- A gravity concentrate dewatering cone and paddock for air drying was incorporated into the plant footprint.
- Copper and gold recoveries to the gravity and flotation concentrate have been updated to reflect 2019 testwork completed on the sulphide and native copper ore blend.
- The tailings thickener diameter has been increased from 42 m to 50 m to handle the finer tailings particle size distribution.

17.3 Process Description

The flowsheet developed for processing ore from sulphide and native copper ores is generally considered to be a relatively standard processing plant design for the treatment of hard rock, copper-bearing sulphide minerals, with the addition of a gravity circuit to recover native copper. All unit operations selected for the plant design are low risk and of proven technology. A commercial analog of the design is being operated at New Gold’s New Afton mill in Canada. The mine plan has allowed for a maximum of 25% of native copper-bearing ores to form the plant feed blend, in this way managing the risks of excessive amounts of native copper in the ore. The design life of the Project is 15 years.

The process description that follows is based on the nominal throughput of 31,200 t/d. The key process units are the following:

- Primary gyratory crushing with indirect dump capability to the crusher, fitted with a rock-breaker unit
- Secondary cone crushing in reverse closed circuit with a secondary screen
- Conveyors moving crushed material to a crushed ore stockpile
- Tertiary HPGR crushing in closed circuit with HPGR wet screens
- Ball milling in closed-circuit with cyclones, which includes a jigging circuit

- Rougher flotation circuit
- Rougher concentrate regrind mill in closed circuit with cyclones and a gravity circuit
- Cleaner, recleaner, and cleaner-scavenger DFR circuit
- Gravity concentrate dewatering and sun-drying paddock
- Flotation concentrate thickening, filtration, and storage
- Gravity and flotation concentrate dispatch
- Tailings thickening and disposal
- Process and fresh water circuits.

The primary crushing station is located near the Little Eva open pit. Primary crushed ore, consisting of 75% sulphide ore and 25% native copper ore, is transferred to the secondary crushing circuit via conveyor belts. The secondary crushing circuit product is stored at the fine ore stockpile, where it is reclaimed to the HPGR circuit. The grinding circuit consists of a ball mill operating in closed circuit with a cyclone cluster. A three-stage jigging circuit is incorporated into the ball milling circuit to process a portion of the cyclone feed and recover coarse native copper particles.

Copper flotation consists of conventional rougher flotation followed by rougher concentrate regrind in a tower mill, and subsequent cleaning by way of a cleaner, recleaner, and cleaner-scavenger DFR circuit, to produce a final flotation concentrate. Gravity concentration is incorporated into the concentrate regrind circuit to process a portion of the regrind cyclone underflow and recover fine native copper particles.

The gravity concentrate is processed through a dewatering cone and further dewatered in a drying paddock by evaporation. The flotation concentrate is thickened using a high-rate thickener prior to filtration in a conventional vertical plate pressure filter. The flotation concentrate is stockpiled in a covered facility prior to transport.

Tailings streams (copper rougher tailings and cleaner-scavenger tailings) are combined and thickened prior to pumping and discharge to the TSF.

Process water is recycled from the tailings thickener overflow and supplemented with process water recovered from the TSF, while overflow solution from the concentrate thickener is reused in the grinding circuit. Make-up water is provided from the fresh water tank. Fresh water is used for gland service for the slurry pumps, for reagent preparation, fire water, filter press cloth wash, and process water make-up.

17.3.1 Primary and Secondary Crushing

The primary and secondary crushing areas of the plant will include the following:

- ROM will be delivered using 141-tonne haul trucks that will feed directly into the crusher dump pocket, with a live capacity of 440 tonnes, and will feed the primary gyratory crusher via apron feeder. The ROM bin live capacity of three truck volumes is larger than typical, and a Copper Mountain Mining Corp. (CMMC) requirement to reduce truck wait times.
- FELs will re-handle ROM ore on the ROM pad as required to ensure crushing circuit availability is achieved.

- The primary crusher will be a gyratory crusher (42 in x 65 in), and will process 1,733 t/h of ROM ore. The gyratory crusher has a feed opening of 1,066 mm, with a closed side setting (CSS) of 130 mm, and with a 450-kW installed power motor.
- Primary crushed ore will feed a double-deck banana screen (4.2 m by 8.5 m) of the same size as the downstream HPGR screens. The secondary screen will be fitted with two screening decks, the first (75 mm) to relieve the load to the lower deck, and the second deck (50 mm) to separate the material that is smaller than the crusher closed side setting and does not require further crushing.
- Oversize material from the secondary screen will feed into the secondary crusher feed bin. The bin capacity is equivalent to 15 minutes residence time. The secondary crusher will be a cone crusher fitted with a 933-kW motor. The cone crusher will operate with a CSS of 50 mm. The secondary crusher product will be returned to the secondary screen, closing the circuit.
- The secondary screen undersize (with P_{80} at 35 mm) will be collected by the fine ore stockpile feed conveyor, which will feed a single, conical fine ore stockpile.

17.3.2 Fine Ore Stockpile

The fine ore stockpile area of the plant will include the following:

- Secondary crushed ore will be fed to the fine ore stockpile, which has a 30,086-tonne live capacity (equivalent to approximately 21 hours). Provision has been made for dozer operations to enable the unit to push material from the dead capacity zone of the crushed ore stockpile into the reclaim area, which may typically occur during extended maintenance outage periods of the primary crusher.
- Fine crushed ore will be reclaimed from the stockpile by two variable-speed apron feeders. Each feeder is designed to provide 100% of the design rate to the HPGR circuit, but the feeders will normally operate at 50% capacity for even draw-down of the stockpile.

17.3.3 HPGR

The HPGR area of the plant will include the following:

- Fine ore will be reclaimed from the fine ore stockpile and transferred to the HPGR feed bin. Feed to the HPGR will be controlled to ensure that the HPGR is choke fed. The HPGR roll dimensions will be 2.4 m diameter by 1.65 m length, with 5.4 MW installed power.
- The HPGR circuit will be closed linked with the grinding circuit, with an availability of 92%, and the nominal crushing rate will be 1,413 t/h.
- The HPGR product will pass to the HPGR screen feed bin, where ore will be fed to two HPGR wet screens (4.2 m by 8.5 m). The top and bottom deck apertures will be 10 mm and 6 mm, respectively. Water will be added to each of the HPGR screen feed boxes and HPGR screens to ensure optimum separation efficiency and ball mill density.
- Screen undersize (with P_{80} of 4 mm) will gravitate to the cyclone feed pump box, while the oversize will return to the HPGR feed bin via the HPGR screen oversize transfer conveyor.

- To prevent damage to the HPGR rolls, metal detectors will be located on the HPGR feeder and on the HPGR screen oversize transfer conveyor that will activate diverter gates to tramp metal bunkers.

17.3.4 Grinding and Jigging

The grinding and jigging areas of the plant will include the following:

- A ball mill operating in closed circuit with a cyclone cluster, producing a cyclone overflow particle size of P_{80} 165 µm. A particle size analyzer will be installed to facilitate the production of ground slurry at the required particle size.
- The HPGR screen undersize gravitates to the cyclone feed pump box, where it is combined with the discharge from the ball mill. Slurry from the cyclone feed pump box is pumped to the ball mill cyclone cluster, with a portion of the flow reporting to the jig circuit. CMMC advised that they preferred an installed standby variable-speed cyclone feed pump, and this has been incorporated into the design.
- CMMC has sourced a competitively priced new ball mill with 7.3 m diameter by 12.2 m effective grinding length, and it will be fitted with a 14.0 MW dual pinion, variable frequency drive. Ausenco has confirmed that this mill will be suitable for the required throughput and grinding duty, and that the ball mill power draw can be controlled using mill speed control should a coarser or finer grind be more economically favourable for certain ore blends.
- Grinding media will be added to the mills as required to maintain grinding efficiency. Ball mill media will be transferred by FEL from the storage bunker into the automated ball mill media system, which will transfer grinding media at a controlled rate to the ball mill feed chute.
- The 12-port cyclone cluster consists of eight 800 mm diameter cyclones operating, and two standby units. This has been designed to operate with a feed density of 52% solids, at 60 kPa, and a circulating load of 300%.
- The two blanks are dedicated to feed the rougher jigs (1 blank per rougher jig). A portion of the cyclone cluster feed (typically 15%) gravitates to the rougher jigs, which consist of two IPJ3500 inline pressure jigs. The rougher jigs produce a concentrate that requires further upgrading, and reports to the cleaner jig. The cleaner jig is a single IPJ2400 inline pressure jig. The cleaner gravity concentrate gravitates to the recleaner jig. Both rougher and cleaner jig tailings gravitate to the cyclone feed pump box.
- The recleaner jig consists of a single IPJ1000 inline pressure jig. Tailings from the recleaner jig report to the recleaner jig tailings tank, from where they are pumped and recirculated to the cleaner jig. Concentrate from the recleaner jig reports to the gravity concentrate dewatering cone.

17.3.5 Flotation and Flotation Concentrate Regrind Circuits

The flotation circuit consists of conventional rougher flotation followed by rougher concentrate regrind, and a cleaner, recleaner, and cleaner-scavenger DFR circuit to produce a final flotation concentrate. The flotation area of the plant will include the following:

- Conventional rougher flotation followed by rougher concentrate regrind, and a cleaner, recleaner, and cleaner-scavenger DFR circuit to produce a final flotation concentrate.

- Six conventional forced-air addition 300 m^3 rougher flotation tank cells. The rougher concentrate flows by gravity to the regrind circuit, and the rougher tailings reports to the tailings thickener via a metallurgical sampler.
- Rougher concentrate is directed to a regrind tower mill with 1,119 kW installed power, operating with circulating load of 150% in closed circuit with a cyclone cluster.
- The regrind cyclone cluster has six ports, with four 400 mm diameter operating cyclones and two in standby. The cyclones operate at 80 kPa, producing a cyclone overflow particle size of P_{80} 48 µm. The regrind cyclone overflow reports to the cleaner flotation head tank.
- A continuous gravity concentrator has been included in the rougher concentrate regrind circuit. The gravity concentrator, treating 25 t/h of solids, has been sized for this application, and processes approximately 16% of the regrind cyclone underflow stream. The resulting concentrate reports directly to the gravity concentrate dewatering cone.
- The cleaner circuit consists of two 18 m^3 DFRs. The cleaner DFRs produce a high-grade concentrate that reports to the final flotation concentrate pump box, while the tailings flow to the cleaner-scavenger DFRs.
- The cleaner-scavenger circuit consists of six 18 m^3 DFRs. Concentrate from the cleaner-scavenger DFRs is pumped to the recleaner DFRs, while the tailings are pumped to the tailings thickener via a metallurgical sampler.
- The recleaner circuit consists of three 6 m^3 DFRs. The recleaner concentrate reports to the final flotation concentrate pump box, and tailings are pumped to the regrind cyclone feed pump box.
- An on-stream analyzer (OSA) and associated multiplexer is included for online process control and sampling.
- Flotation collector (PAX) is added to the ball mill and rougher and cleaner flotation. Frother (MIBC or Polyfroth H27) is added to the rougher, cleaner, and cleaner-scavenger flotation as required. Sulphidizer (NaHS) is added to the two last rougher flotation cells.

17.3.6 Gravity Concentrate Handling

The gravity concentrate handling area of the plant is summarized as follows:

- A 1.8 m diameter gravity concentrate dewatering cone receives the concentrate from the recleaner jig and Knelson concentrator. The dewatering cone overflow solution is recovered and sent to the flotation concentrate thickener. Gravity concentrate solids settle for collection at the underflow cone at a density of 70% solids.
- The dewatering cone underflow gravity flows to the gravity concentrate paddock, where the remaining moisture will evaporate.

17.3.7 Flotation Concentrate Handling

The flotation concentrate handling area of the plant is summarized as follows:

- A 16 m diameter high-rate concentrate thickener, including a Frothbuster system to minimize foam and concentrate loss, is included in the design. The concentrate thickener underflow density has been specified as 65% solids. Flocculant is added to facilitate settling and limit suspended solids in the supernatant solution. Overflow solution from the concentrate thickener is recycled as process water to the primary cyclone pump box to utilize the presence of residual flotation reagents.

- The thickener underflow stream is pumped to an agitated filter feed tank by a peristaltic pump. The concentrate storage tank retention time is 24 hours.
- The design specification is for an automatic pressure filter (horizontal filter press) with a filtration area of 140 m^2 (using sixty-two 1.5 x 1.5 m plates) and air drying, to achieve a concentrate filter cake with 9% moisture, operating at an availability of 65%, with a cycle time of 12.5 minutes.

17.3.8 Concentrate Storage and Load-Out

Gravity and flotation concentrates will be stored and transported separately:

- A covered building provides storage for up to 2,400 tonnes of flotation concentrate (equivalent to three to four days of production at design rates) and a FEL is used to optimize concentrate storage within the building. Flotation concentrate is loaded into trucks by FELs. Trucks are positioned on a scale prior to loading. The scale provides feedback to the FEL operators as the trucks are filled. When trucks are full, they drive through a wheel wash, and then the concentrate is transported to an off-site facility in half-height sealed containers.
- The gravity concentrate paddock provides storage of up to 200 tonnes of gravity concentrate. The gravity concentrate paddock has two compartments, each with capacity of 100 tonnes. While concentrate fills one compartment during a period of 11 days, the gravity concentrate in the other compartment dries out and is loaded into a truck by a FEL, similarly to the procedure described for the flotation concentrate. The proportion of copper production as gravity concentrate is anticipated to vary significantly over the life of mine; therefore, the residence time provided by the drying paddocks is estimated based on the nominal mill feed blend composition and corresponding testwork results.

It should be noted that, due to potential road closures during the wet season, the nominated three to four days storage may not be sufficient to ensure continuity of production. It is possible that additional concentrate storage capacity could be realized by using shipping containers designed to transport concentrate.

17.3.9 Tailings Handling

Rougher and cleaner scavenger tailings flow by gravity to the tailings thickener feed box and gravitate into the tailings thickener. Flocculant is diluted in the static mixer to 0.1% w/w, and then added to the thickener feed well to assist with solids settling and to maintain overflow clarity, suitable for recycling and reuse in the process plant.

The tailings thickener is designed with a diameter of 50 m to accommodate the nominal throughput of 31,200 t/d and particle size P_{80} of 165 µm.

The tailings thickener supernatant overflows to the process water tank. The thickener underflow is pumped by duty and standby tailings pumps to the TSF. The associated discharge overland pipeline to the TSF is fitted with dual flow meters for leak detection.

The TSF and the associated distribution piping is designed for the subaerial deposition of tailings and the associated recovery of decant solutions, and this facility is equipped with under-drainage leak detection.

Other aspects of the 2018 Feasibility Study tailings thickener area that have been revised include:

- The tailings thickener was originally designed to produce a flocculated underflow density of 58% solids, with the thickener overflow recycled to the process water system. This has since been revised to an underflow density of 63% solids.
- The thickener underflow will be pumped to the TSF as final tailings by duty and standby tailings pumps connected to the thickener rather than to an underflow hopper.

17.3.10 Reagent Handling and Storage

Based on the earlier metallurgical testwork programs, the recommended reagent scheme for the Eva Copper Project was determined to include:

- Potassium amyl xanthate (PAX) collector
- Alkyl dithiophosphate promoter Aero A3477 (or RTD1481 equivalent)
- Frother reagent methyl isobutyl carbinol (MIBC) or Polyfroth H27 equivalent
- Sulphidizing reagent sodium hydrosulphide (NaHS) will be required to treat the blend of sulphide and native copper ore.

The alkaline nature of the slurry (pH value around 7.7) was found to be suitable for the copper flotation process, and therefore no pH modifier will be required. The slurry alkalinity also allows for the addition of PAX to the grinding circuit with no resultant loss of the PAX that would occur under acidic slurry conditions.

It is possible that some modifications to the reagent preparation section may also be required for ores from the satellite deposits. A test reagent preparation circuit has been included in the design, which can be used to evaluate newer and/or more efficient flotation reagents on different ore types.

Flocculant will be used in both the concentrate and tailings thickeners to aid the settling process. An appropriate flocculant preparation circuit has been included in the design.

Subject to the selection of a vendor to supply the various reagents, and possible changes to the design that may take place as a result, the following describes the reagent preparation section at the present time.

The chemical reagents PAX and MIBC (or Polyfroth H27), identified in the testwork programs, are added to the slurry streams to facilitate the recovery of the copper sulphide minerals into a final concentrate. Fresh water is required for making up the PAX reagent to the specified solution strengths. Preparation of the various reagents require the following:

- Dangerous goods reagent storage shed
- Raw water supply
- Handling system for bulk reagent containers

- Reagent handling, originally based on preparing two batches per day based on reagent usage specified in the design criteria
- The PAX mix tank produces the required solution strength; the prepared solution is then transferred to the storage tank for periodic distribution to the day tank, which supplies the reagent for addition to the respective process streams at the required dosage rates
- PAX is supplied in 1-tonne bulk bags; the mix tank produces a 20% strength solution
- MIBC (or Polyfroth H27) frother is supplied as a 100% stock solution in 20-tonne tanker loads and added without dilution
- Sulphidizer (NaHS) is supplied at 70% strength in 20-tonne tanker loads and will not require dilution prior to addition to rougher flotation cells
- flocculant is supplied in 1-tonne bulk bags; the mix tank produces a 1% flocculant solution for distribution to both thickeners.

To ensure spill containment, the reagent preparation and storage facility is located within a containment area designed to accommodate 110% of the solution content of the largest tank. The reagent tanks are equipped with level indicators and instrumentation to ensure that spills do not occur during normal operations. Appropriate ventilation and fire and safety protection are provided. Each reagent line and addition point are labelled in accordance with Workplace Hazardous Materials Information Systems (WHMIS) standards, or equivalent. All operations personnel receive WHMIS training, along with additional training for the safe handling and use of reagents.

As a result of potential road closures due to flooding, mines in North West Queensland typically hold several weeks' stock of reagents and grinding media ahead of the wet season (i.e., build-up stock by November). This should be reviewed in the next phase of the Project to ensure that there is sufficient reagent storage capacity to avoid potential production interruptions.

17.3.11 Assay and Metallurgical Laboratory

The assay laboratory is equipped with the necessary analytical instruments to provide all the routine assays for the mine, processing plant, and the geological and environmental departments. The most important items of equipment will include the following:

- Fire assay equipment
- Atomic absorption spectrophotometer (AAS)
- X-ray fluorescence spectrometer (XRF)
- LECO furnace.

The metallurgical laboratory will undertake the necessary testwork to monitor the metallurgical performance of the plant and improve flowsheet unit operations and efficiencies, while also undertaking testwork on the satellite deposits slated for future mining. This facility is equipped with laboratory crushers, ball and stirred mills, particle size analysis sieves, flotation cells, filtering devices, and balances.

17.3.12 Site Services

The following is a summary of the utilities that will be installed to service process plant operations:

- A process water tank to receive the overflow solution from the tailings thickener for redistribution in the plant
- Piping to return concentrate thickener overflow solution to the primary cyclone pump box to maximize reagent reuse
- An HDPE-lined process water pond with a design capacity of 30,000 m^3, and associated duty and standby pumps and distribution piping
- An HDPE-lined fresh water pond with a design capacity of 18,000 m^3, and associated duty and standby pumps and distribution piping
- A potable water treatment plant consisting of ultra-violet (UV) sterilization and reverse osmosis (RO) filtration
- A potable water tank and associated duty and standby distribution pumps and piping to supply site ablutions, safety showers, crib rooms, and the accommodations camp
- Plant and instrument air systems, including air compressors, air receivers, and air dryers; air blowers will supply low pressure air to the flotation cells
- A programmable logic controller (PLC)-based process control system (PCS), together with a supervisory control and data acquisition interface and control room
- Uninterruptible power supply (UPS) system to maintain the process control system in the event of a power failure
- An emergency power supply for the following equipment in the event of a power failure: fire water system, both thickener drives, the tailings thickener underflow pumps, the concentrate thickener underflow pumps, and the agitators in the process plant
- Other equipment may be identified for security and/or safety reasons during the next phase of the study:
  - Closed-circuit television (CCTV) units to assist with plant monitoring and security
  - Communication availability via 4GX mobile phone coverage and fibre optic cable, provided by Telstra as part of its service delivery to the Dugald River Mine Project.

17.3.13 Water Supply

Two separate water supply systems are required to support the operation: one for fresh water, and the other for process water.

The fresh water is supplied to the fresh water / firewater tank from the fresh water supply source identified in the earlier studies. Fresh water is primarily used for the following:

- Fire water for emergency use
- Mill cooling water
- Gland water for the slurry pumps
- Reagent make-up water
- Filter press cloth wash
- Potable water supply.

The fresh water / firewater tank is equipped with a standpipe that will ensure that the tank is always holding at least 500 m³ of fresh water for supply of fire water.

The overflow of the fresh / firewater tank reports to the fresh water pond, from where fresh water make-up can also be pumped by two fresh water barge pumps (one operating, one standby) in an emergency event should water from the fresh water tank not be available.

The potable water from the fresh water source is treated and stored in the potable water storage tank prior to delivery to the various service points.

The process water supply system consists of two separate circuits. The concentrate thickener overflow water is returned directly to the cyclone feed pump box for reuse in the classification and grinding circuit. The tailings thickener overflow solution is returned to the process water tank for reuse in other parts of the plant, such as spray water for the ore stockpile, SAG mill dilution, SAG mill discharge screen, ball mill trommel screen spray water, cyclone feed make-up water, jigging hutch water, and flotation and flocculant dilution water. The process water tank is connected to the process water pond, which provides surge capacity during the processing operations. Plant water make-up comes from the fresh water tank.

17.3.14 Air Supply

Separate low-pressure and high-pressure air service systems supply air to the following areas:

- Low-pressure air for all the flotation cells is provided by two multi-stage centrifugal blowers (operating as duty and standby).
- High-pressure air is provided for the plant and concentrate filter press from three plant air compressors. Compressed air is dried prior to storage in the instrument air receiver, from where it is reticulated around the plant.
- High-pressure air is provided for the primary and secondary crushing areas using a dedicated air compressor. Compressed air is dried prior to storage in the primary/secondary crushing air receivers, from where it is reticulated around the primary/secondary crushing plant.
- High-pressure air is provided for the HPGR and secondary screening areas using a dedicated air compressor. Compressed air is dried prior to storage in the HPGR/secondary screening air receivers, from where it is reticulated around the HPGR/secondary screening plant.

17.3.15 Online Sample Analysis

The plant will rely on the automatic sampling and analysis of six online streams. This system provides the necessary information for process control, and the samplers also take sufficient sample quantities to use for checking/standardization and possible metallurgical testwork purposes. The six process streams that are sampled continuously are the following: rougher feed, rougher concentrate, rougher tailings, cleaner-scavenger tailings, and concentrate thickener feed.

The information obtained from these samplers enables overall recoveries and grades of all the process streams to be calculated, thereby providing an overall process performance balance of the plant. The analysed and excess sample slurries are collected in the sample return tank, and returned to the slurry feed stream to the rougher flotation circuit.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

The Eva Copper mine, plant, and associated open pits are located 76 km northwest of Cloncurry. The site can be accessed by way of the sealed (paved) Burke Development Road, and a planned site access gravel road of approximately 8 km.

Infrastructure required to be installed to support the operation includes:

- Roads: main access road, plant site, tailings storage facility (TSF) light vehicle track, explosives and emulsion access road, Cabbage Tree Creek (borefield light vehicle track, and haul roads
- Security office and tag in/out board building
- Administration building, training, first aid, plant crib, and car park
- Control room (primary crusher and rock breaker)
- Control room (grinding area)
- Processing plant office
- Concentrate storage shed and weighbridge
- Reagent storage and building
- Assay laboratory and sample preparation area
- Communication facilities
- Mining infrastructure
- Mine change house
- Truck shop, plant workshop, warehouse, and office
- Tire services pad and services area
- Lubricant storage
- Hydraulic hoses storage
- Fuel storage and dispensing
- Borefields (Little Eva pits and Blackard dewatering wells and Cabbage Tree Creek supply)
- Overland HV transmission line from the tap near Dugald substation (11 km)
- Fresh water supply and treatment
- TSF (424 ha)
- Site sediment management installations
- Creek diversion channel around Little Eva and other pits and surface water bunding
- Explosive bulk storage depot and magazine
- Emulsion facility
- Accommodation village and associated infrastructure.

Figure 18-1 illustrates the broader site infrastructure layout.

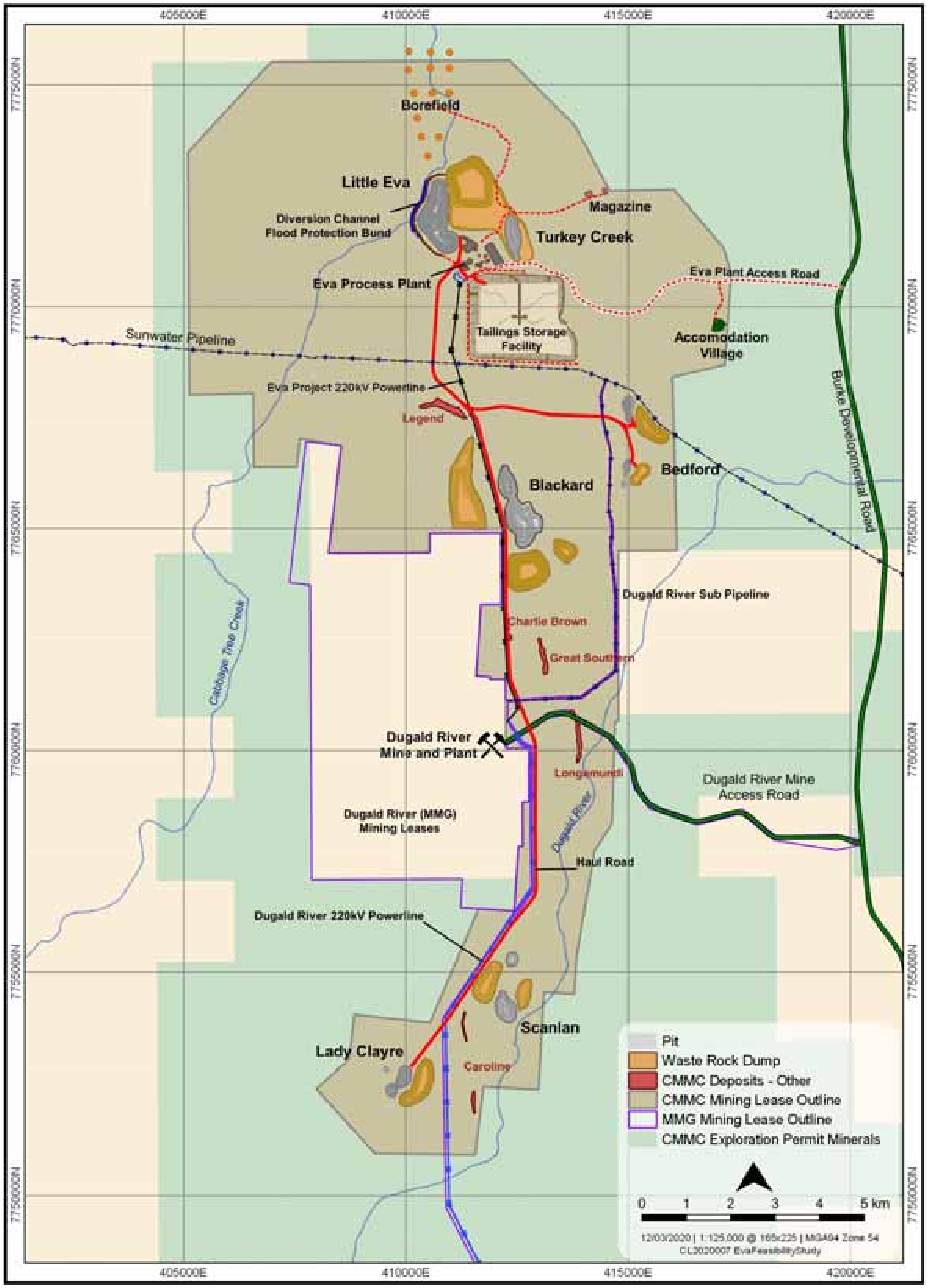


Figure 18-1: Infrastructure Layout

18.1.1 Existing Regional Infrastructure

The closest regional centre to the Project area is the town of Cloncurry. At the 2016 census, the town had a population of 2,719. Cloncurry is located 770 km west of the city of Townsville. The Burke Developmental Road runs from Cloncurry in a generally northerly direction past the Project leases to the northern town of Karumba. The road passes the Project area 8.5 km to the east of the proposed location of the Little Eva pit. Other major regional infrastructure includes HV lines transmitting power from the Mica Creek/Diamantina power station in Mount Isa through Cloncurry to the Ernest Henry mine, and the Lake Julius water pipeline crossing the Eva Copper site 2 km south of the planned processing plant (

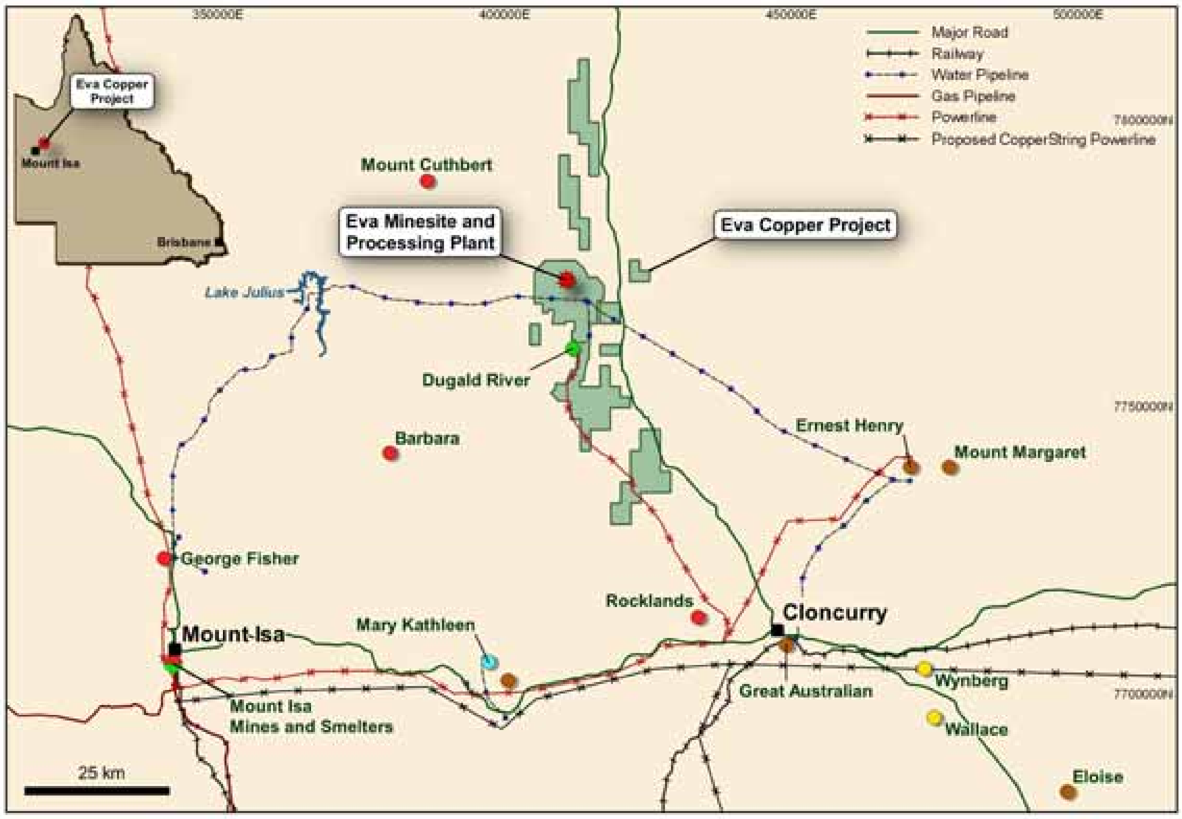


Figure 18-2). An HV transmission line has been installed by MMG to service the Dugald River Mine, located 11 km to the south of the proposed Little Eva plant.

The site is located 76 km northwest of Cloncurry airport, 194 km northeast of Mount Isa airport, and 1,500 km northwest of Brisbane. Existing minor site infrastructure includes access tracks for drilling activities, and station roads to service the local cattle grazing industry.

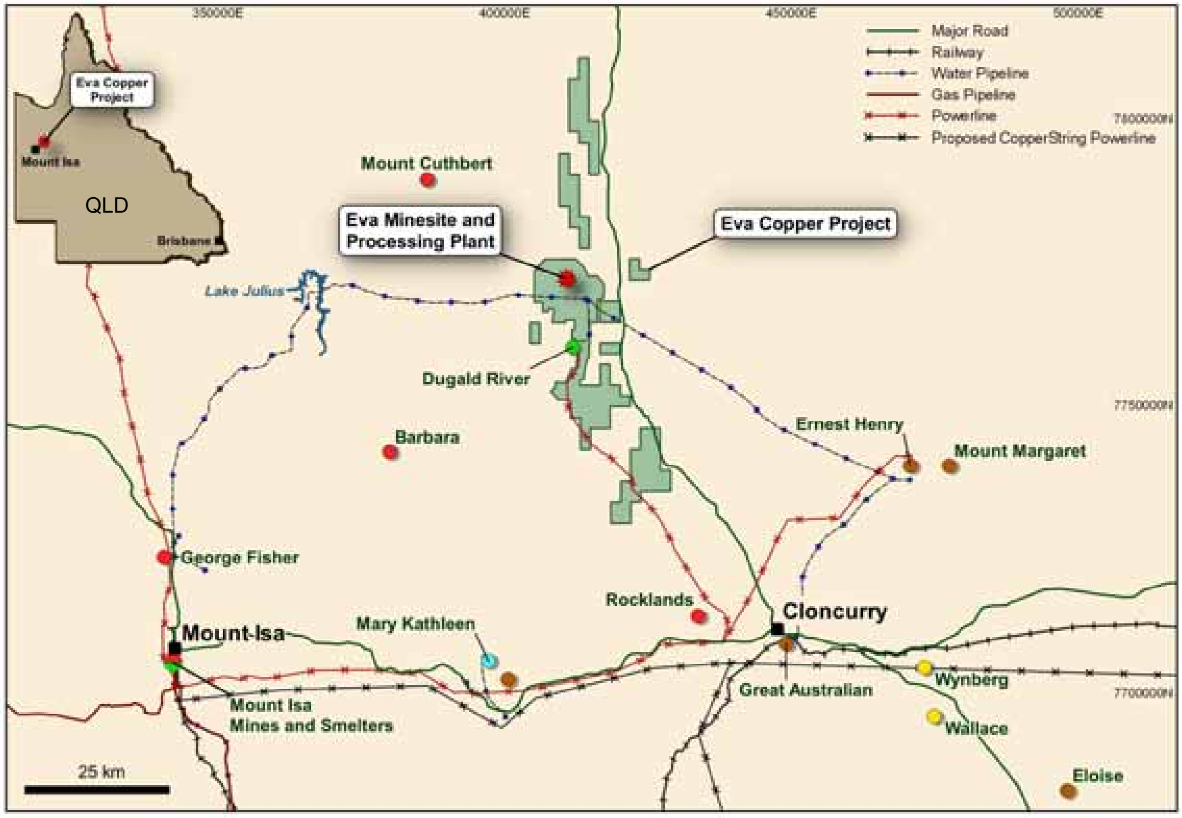


Figure 18-2: Location of the Project, the Little Eva Plant, and Regional Infrastructure

18.2 Project Site Infrastructure

18.2.1 Access and Roads

18.2.1.1 Processing Facility and Accommodation Village Access Roads

The new site access road would be established from the Burke Developmental Road to the accommodation village and the processing plant. A T-junction with traffic merge lanes would be constructed on the Burke Developmental Road approximately 70 km north of Cloncurry. The site access road is designed to traverse in a general westerly direction for 8.5 km. At the 2.8 km point, a T-junction occurs, which leads to a 900 m long access road servicing the accommodation village. The road continues another 5.7 km west to the Eva processing facility and administration complex.

The main access road will be constructed as a two-lane, gravel road, each lane 4 m wide, with 1 m wide shoulders on each side. The road base course will be compacted sandy/clayey gravels, 150 mm to 300 mm thick, which will be transported to the route from borrow pits located within 1 km from the work front. Construction water for the access road will be sourced from dewatering bores, temporary dams, and pumps located around the Little Eva pit. Water will be delivered to the road construction work fronts by water trucks provided by the mine.

Plant roads will be formed and constructed by the process plant earthworks contractor. Mine haulage and internal roads will be constructed by the Company during site establishment. The route of the processing facility and camp access road in Figure 18-1 has been selected to minimize creek and river crossings, so as to maintain access during the wet season and flood events. There will be only one creek crossing on the currently proposed 8.5 km access route, which will have culverts installed designed for a 1-in-50-years rainfall event. This approach to access is included in the current cost estimate for the Project.

18.2.1.2 Bulk Storage Facility and Magazine and Emulsion Facility Road

The explosives depot and magazine road will be constructed as a two-lane gravel road, each lane three metres wide, with one metre wide shoulders on each side. The road base course will be compacted sandy/clayey gravels, 150 mm topping gravel over 450 mm base gravel material. The road distances will be from the plant to the emulsion facility, 2.9 km long, and from the emulsion facility to the explosives magazine, 0.3 km long.

18.2.1.3 Tailing Storage Facility Light Vehicle Track, CTC Borefield Light Vehicle Track, and Power Line Corridor

Access tracks to the TSF, CTC Borefield, and power supply power line will be built. These access tracks will generally be 6 m wide tracks constructed by forming up the existing ground and establishing drains. Some imported fill will be used where required by ground conditions. Floodways, spoon drains, and some culverts will be installed at waterway crossings. Allowance for the following access tracks has been made in the capital cost estimates:

- 11 km long power line corridor access tracks
- 4 km long TSF light vehicle access tracks
- 4.8 km long CTC borefield light vehicle access tracks.

18.2.2 Site Development and Drainage

Site surface water management will consist of diversion channels upstream of the TSF to divert clean runoff around the TSF, and divert sediment-laden runoff from the TSF embankment downstream face into sediment control structures.

A process water pond and diversion drains will be placed around the plant site. Diversion drains around the plant site will minimize the catchment area for the plant site drainage pond. The process water pond will collect potentially contaminated runoff from the plant area. The pond will be lined and fitted with a barge and submersible pump to transfer water from the pond back into the plant process water system.

Two sediment containment ponds (SCP 1 and SCP 2) will be constructed to accommodate the design storm event without overtopping. The SCPs were sized with spillways to accommodate the design storm event. SCP 1 will be located south of the plant site (on the TSF's northwestern corner), and will have a design capacity of 180,000 m³. SCP 2 will be constructed to the north of the Little Eva waste dump, and will have a design capacity of 48,000 m³.

The SCP 1 and SCP 2 decant system will consist of a diesel-powered centrifugal dewatering pump with a floating "skimmer" pipe, which will sit on the water surface, draining water from the top of the storage into a solids discharge pipe, which will run on top of the embankment. The purpose of the SCPs will be to capture sediment and ensure that overflow water is sufficiently clean for release to the local watercourses or plant process water tank.

The process water pond, also called the plant site drainage pond, will be an HDPE-lined pond 89 m by 89 m at the top, nominally 5 m deep, with a design capacity of 30,000 m^3. The pond will be fitted with a submersible transfer pump and pipeline (with an outside diameter (OD) of 250 mm) capable of transferring the contents of the pond to the process water tank within 72 hours.

Plant site bulk earthworks will consist of clearing and grubbing of the site, removal to stockpile of topsoil, profiling of the site to ensure cross-fall drainage to the plant site drainage pond to the west of the plant, and construction of hardstands and earthworks foundations for major equipment.

18.3 Power Supply

The plant and infrastructure electrical system will be designed and installed to comply with all relevant standards and statutory requirements to provide high reliability and ease of maintenance in accordance with Queensland standards. With 42.5 MW of installed drives, the average power draw for the processing plant during operations will be approximately 30.7 MW.

Power for the processing plant will be supplied from gas-fired generators in Mount Isa, at either the Mica Creek power station or the Diamantina power station. Gas supply to these stations is provided by the Carpentaria Gas Pipeline. Power is transmitted along the North West Power System (NWPS) 120 km to the network operator's 220 kV Chumvale substation, adjacent to the town of Cloncurry. From Chumvale, the power is transmitted along MMG's 64 km long, 220 kV, Dugald River overhead transmission line, terminating at MMG's Dugald River substation. A tap will be installed adjacent to the MMG site, and an 11 km extension will be constructed to supply power to the step-down substation (220 kV to 11 kV) at the Eva Copper Project plant site, from which power will be distributed throughout the process plant and to site infrastructure.

The Project has a commercial understanding for access on the MMG Dugald River 220 kV line at the Eva Copper Project demarcation tap point. In addition, transmission line maintenance costs have been provided for in the operating cost power calculations.

For this study, the cost of power at site will be US$0.1211/kWh (AU$0.1877/kWh) for the first three years of plant production, based on a power transmission from Mount Isa. From year four onwards the cost of power should be US$0.0635/kWh (AU$0.0985/kWh) based on a term sheet with CopperString, the proponent for developing a high voltage electricity transmission line to connect electricity users in the North West Minerals Province (NWMP) and the Mount Isa region to the National Electricity Market (NEM) at Woodstock near Townsville.

On-site emergency backup diesel generator power will be provided to supply plant essential services, cafeteria, and accommodation services.

18.4 Water

18.4.1 Supply and Treatment

Raw water for the Project will be supplied from a borefield to be established at Cabbage Tree Creek, located approximately 2 km north of the Little Eva pit. CMMPL completed a hydrogeological investigation to define the borefield capabilities. Following the successful exploration program, two test bores and a monitoring bore were constructed (CTPB01 through CTPB03), which will eventually be expanded to form the Cabbage Tree Creek borefield. Test pumping conducted at two of the three test bores (CTPB02 and CTPB03) confirmed supply at a rate of 25 m^3/h to 50 m^3/h each. An additional 12 wells will be added for a total of 15 wells in the Cabbage Tree Creek borefield. The wells will be powered by an 11 kV distribution line. Each well has been calculated to produce an average of 6.5 L/sec for a total of 351 m^3/h. The water then will be pumped into a 1,575 m^3 nominal capacity collection tank at the western side of the Little Eva pit on elevation, which will be pumped via centrifugal transfer pumps, as and when required, to the fresh water/firewater tank, which is 14.8 m in diameter with a height of 9.2 m (capacity 1,575 m^3) located in the processing plant.

Mine dewatering of the Little Eva pit will be accomplished by ten dewatering well holes equipped with submersible pumps, strategically located around the pit to ensure that mining remains as dry as practical. It is calculated that the pit dewatering wells and pit dewatering pumps will produce a total of 180 m^3/h of raw water. The wells will each be designed for a capacity of 18.0 m^3/h, and will each pump through a dedicated 180 mm OD HDPE DR13 pipeline into the collection tank at the western side of the Eva Pit. The wells will be powered by an 11 kV distribution line. The individual pipelines from the wells to the common water tank will be run above ground, except at road crossings.

Return water from the TSF will also be used to supply process water. A system has been designed using three 75 kW submersible pumps, each capable of pumping 5,443 m^3 of reclaim water per day, to a maximum of 17,136 m^3/d using from one to three pumps. The reclaim water will be transferred by a 2 km long HDPE pipeline to the process water tank. It will also be possible to source water from the Lake Julius to Ernest Henry pipeline owned by SunWater. Potable water for the accommodation village will be supplied from a water well to a tank, then to a water treatment plant. Potable water for the plant site will be treated prior to plant entry in a reverse osmosis (RO) water treatment plant. The safety showers will be supplied from a 23 m^3 chilled water tank, and distributed around the plant site using duty or standby pumps via a ring main.

18.4.2 Water Management

Key issues for the Project in relation to water management are:

- Diversion of surface water around the open pits and site infrastructure
- Dewatering of the Little Eva pit and other subsequent pits
- Supply of raw and fresh water for ore processing and site services.

The Project is situated close to the headwaters of the Dugald River, a tributary of the Cloncurry River. Cabbage Tree Creek, a tributary of the Leichhardt River, meanders in a generally northerly direction adjacent to, and immediately west of, the proposed Little Eva open pit (Figure 18-3). Both Dugald River and Leichhardt River drain northward, eventually discharging into the Gulf of Carpentaria. The headwaters of these rivers lie in moderately dissected terrain, with minor tributaries ending in steep, stony escarpment remnants of an older plateau surface. Higher land levels range from 356 mASL at Mount Maggie at the headwaters of the Dugald River, to 313 mASL at the headwaters of Cabbage Tree Creek. Mount Rose Bee in the central part of the Project is 285 mASL in elevation. The bed of Cabbage Tree Creek near Little Eva is 157 mASL in elevation. General relief across the Project area is between 100 mASL to 128 mASL. The hills are aligned in a generally northerly direction, controlled by the regional strike of the Proterozoic rocks.

The Project is subject to the *Water Act* 2000, and is specifically located within the subordinate legislation of both the Water Plan (Gulf) 2007, and the Water Plan (Great Artesian Basin and Other Regional Aquifers) 2017.

The key issues for the Project in relation to surface water management relate to maintaining access to site and constructing an adequate surface water diversion infrastructure to ensure that Cabbage Tree Creek water flows do not enter the Little Eva pit. Similar smaller diversions or bunds are required for Blackard Creek and Dugald River for the Blackard and Scanlan deposits respectively.

The planned Project access road from the Burke Developmental road will be engineered to withstand periodic flood events and properly manage water flows by utilizing appropriate rock-lined drainage under pavement culverts and spillways. The detailed access road design will be carried out during the next engineering phase of the Project.

Cabbage Tree Creek is an ephemeral waterway that generally remains dry, except during the wet season from December through March. Cabbage Tree Creek will occasionally flood for short durations during these months, with flood waters dissipating quickly, leaving local water holes along the length of the creek.

The mining pushback on the western side of the ultimate Little Eva pit will intersect Cabbage Tree Creek at approximately 500 m down from surface. To ensure water flow is diverted away from the pit, an 800 m long Cabbage Tree Creek diversion, consisting of a channel and flood bund, has been designed by Knight Piésold. The diversion intersects the existing creek at the southwest of the pit, and diverts the water around the western edge of the pit, discharging it into the existing waterway to the northwest of the pit. Construction of this diversion has been approved as part of the 2016 EA for the Project.

In the initial years of the Project, process water will primarily be provided by Little Eva pit dewatering and the borefield established at Cabbage Tree Creek. A portion of this water will be treated via RO filtration to provide the potable water requirements for the Project. Return water from the TSF will also be utilized by the process. Except for a short period prior to plant commissioning, the Project is not expected to generate excess water. There is no requirement to discharge collected water to the natural environment.

Studies on pit dewatering and groundwater supply were carried out in 2011 by both Morgan and Rockwater, and again in 2018 by Rockwater. Knight Piésold (KP) carried out studies for inundation and flood modelling based on the 2012 Definitive Feasibility Study (DFS) mine layout, and subsequently revised the work in 2016 with inclusion of the Turkey Creek pit, which required the relocation of the TSF. The revision included updated feasibility designs for the Cabbage Tree Creek diversion and regional site surface water management structures. In 2020, KCB revised the TSF design to accommodate the increase in storage capacity required by the increased mine plan. KCB included an assessment of settling densities, and a high-level water balance to estimate expected return water recycle rates to the process plant under a range of climatic conditions.

Dewatering and yield tests have yet to be carried out on the Turkey Creek, Bedford, and Lady Clayre pits, and although they are expected to contribute some water to the process water balance, they have not been considered. Any contribution from the satellite pits will reduce the requirement for raw water from the Cabbage Tree Creek borefield.

This section describes aspects of water supply and surface water and groundwater management for the mine, process plant, and infrastructure.

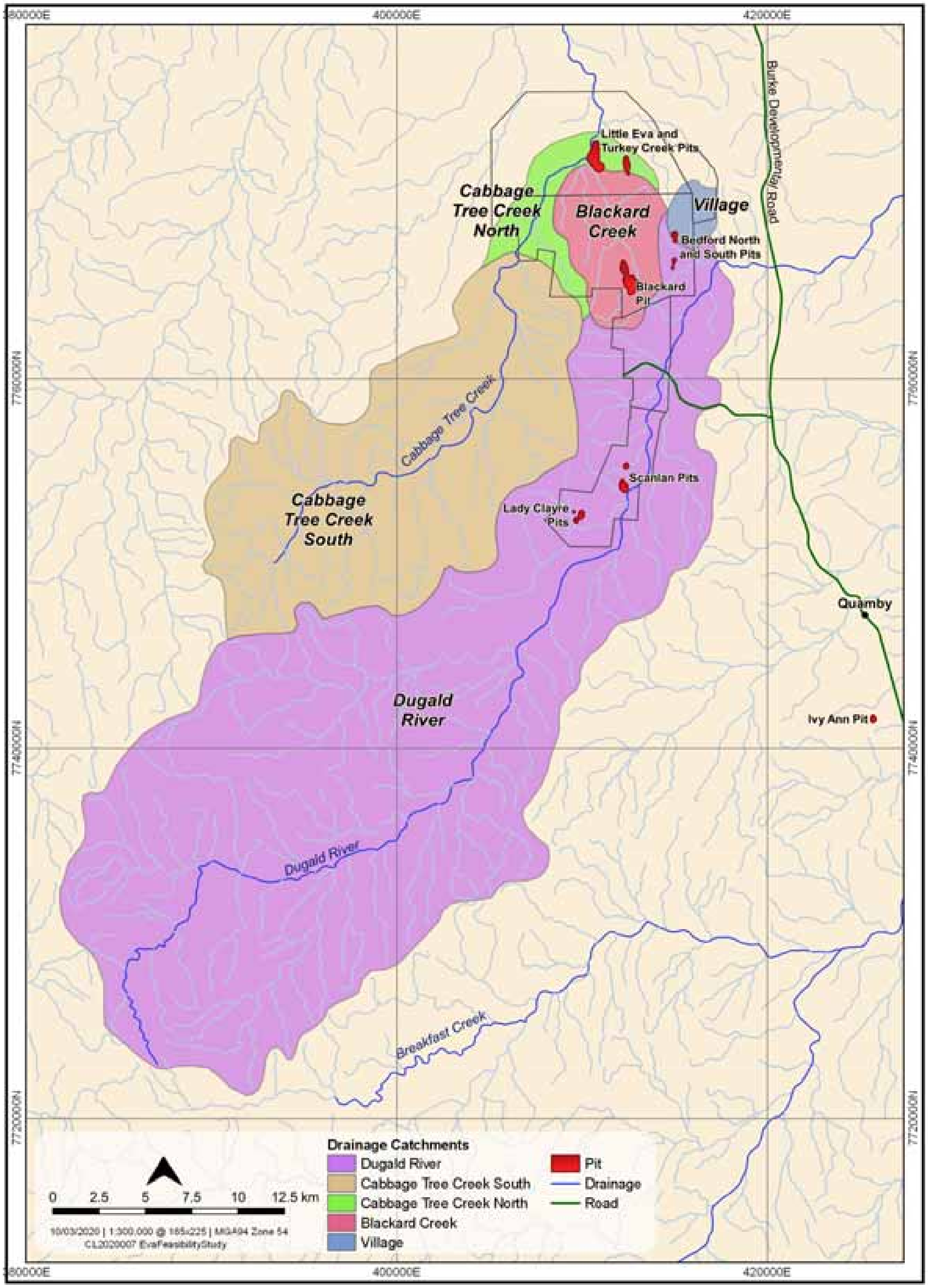


Figure 18-3: Regional Water Drainages and Catchments

18.4.3 Plant Site Surface Water and Sediment Management

Key aspects of site surface water management initiatives shown in Figure 18-4 are the following:

- Diversion channels upstream of the TSF will divert clean runoff around the TSF where possible, or the water will be captured and pumped into the TSF where surface topography restricts the development of diversion drains.
- Sediment-laden runoff from the TSF embankment will be captured into collection drains and conveyed to sediment control ponds.
- Diversion drains around the plant site will report to a plant site drainage pond. Any water entering the operations area of the plant site is deemed to be contaminated water, and must be collected. For this reason, it is important to divert as much runoff as possible prior to it entering the plant site area. Diversion drains around the plant site minimize the catchment area for water reporting to the plant site drainage pond. The pond will be lined and fitted with a submersible pump to transfer water from the pond into the plant process water tank.

The site surface water management includes runoff from the TSF downstream embankment, which will be captured into collection drains and conveyed to sediment control ponds. An important part of Project earthworks will be the provision of suitable erosion and sedimentation control measures to minimize impacts from the Project to downstream environments.

Sediment ponds (SCP) were located based on pit layouts, proposed waste dump locations, and other infrastructure. The SCPs were sized with spillways to accommodate the design storm event. The SCP decant system consists of a floating skimmer pipe, which sits on the water surface, draining water from the top of the storage into a discharge pipe. A valve will be situated downstream of the SCP embankment to allow release of water if it is deemed suitable for discharge.

The SCPs are categorized as Low Hazard Dams according to published Queensland guidelines.

The plant site drainage pond will be an HDPE-lined pond, 89 m by 89 m at the top, nominally 5 m deep, with a design capacity of 30,000 m^3. The pond will be fitted with a submersible transfer pump and pipeline (250 mm OD) capable of transferring the maximum volume of the pond to the process water dam within 72 hours.

18.4.4 Cabbage Tree Creek Diversion Channel

The Cabbage Tree Creek diversion structure was redesigned by Knight Piésold in 2019 and provides flood inundation protection on the west side of the Little Eva open pit. The diversion has been designed to environmental permitting level, and its function is to divert water flows from the creek around the perimeter of the open pit and, as mining progresses, it will eventually encroach into a section of the current alignment of the waterway. The design as illustrated in Figure 18-4 and Figure 18-5 is included in the approved Project EA.

To determine the design parameters, Knight Piésold undertook site reconnaissance and carried out flood and inundation modelling for the Cabbage Tree Creek catchment, which was assessed to be an area of 334 km^2 upstream of the Project location.

A Hydrologic Engineering Center's Hydraulic Modeling System (HEC-HMS) hydrological model was used to evaluate the water flow characteristics, using the Probable Maximum Precipitation (PMP) and 20-year Average Recurrence Interval (ARI) storms at a number of potential durations. Smooth curves were plotted through the data pairs (storm duration, peak outflow) from the model output for the downstream end of the model. The maximum inflection points on the resulting curves defined the critical durations for the assessed design storms near the Little Eva Pit. On this basis, the critical duration for PMP storms was determined to be 120 hours, and the critical duration for 20-year ARI storms was determined to be 36 hours.

The proposed design envisions a combination of a low-flow channel to manage most Cabbage Tree Creek flows, and an elevated diversion bund designed for seasonal periodic flood events. A Cabbage Tree Creek RiverFLO-2D inundation model was executed with the two critical duration storms (120-hour duration PMP, and 36-hour duration 20-year ARI). Peak flow rates from the hydrographs of multiple regional catchments were used, along with the assumptions to determine the peak inundation resulting from each storm. Water surface elevation information extracted from the RiverFLO-2D model under peak PMP inundation conditions was used to establish the necessary crest profile of the proposed diversion bund required to contain the predicted inundation. Channel invert elevations were then designed along the planned alignment of the proposed low-flow channel, which achieved both the predicted inundation performance and hydraulic velocity requirements. Hydraulic velocity results extracted from the RiverFLO-2D model were then used to size rip-rap revetment for protecting the diversion bund from scour during the assessed flood events.

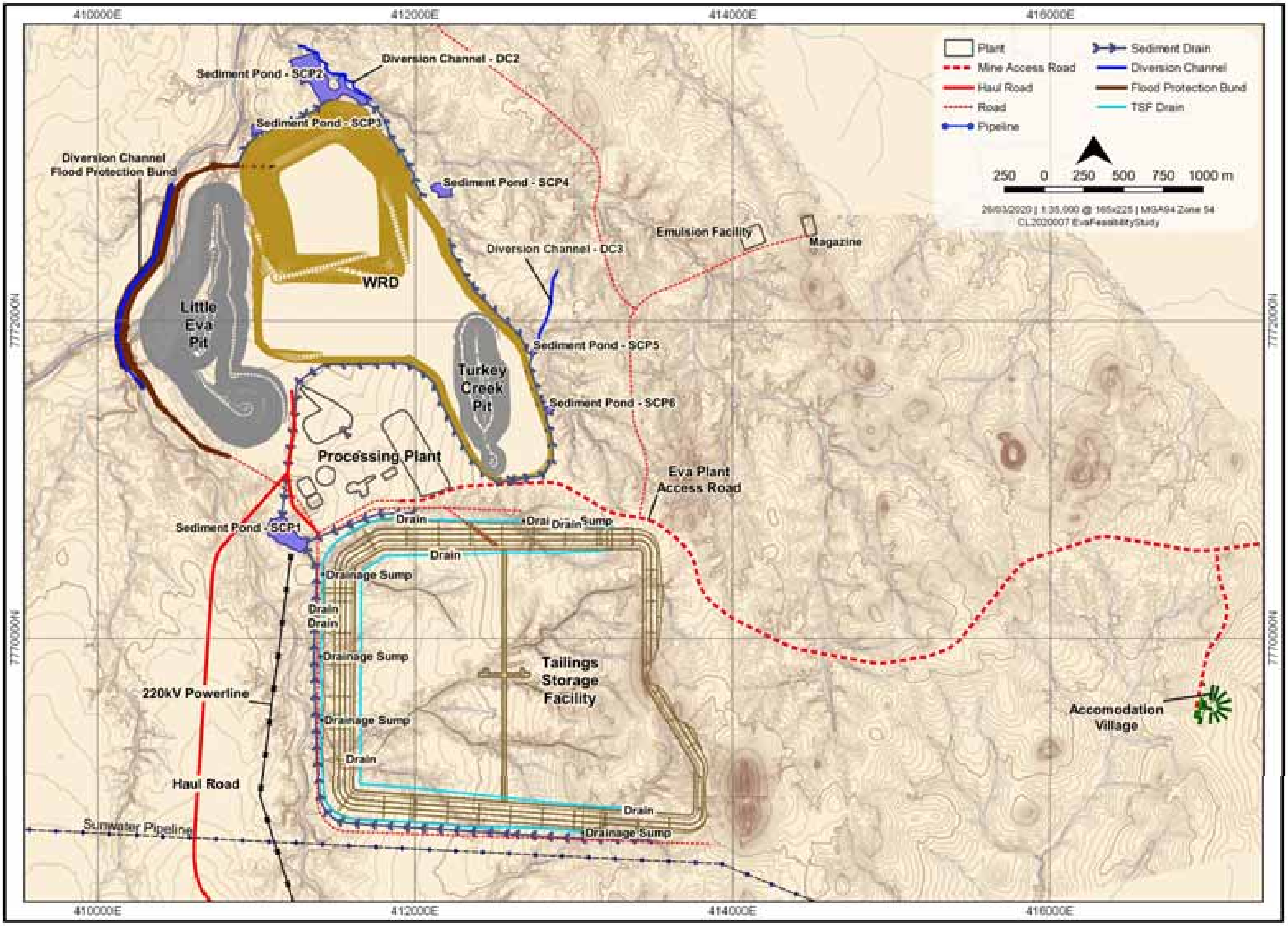


Figure 18-4: Site Surface Water Management Plan

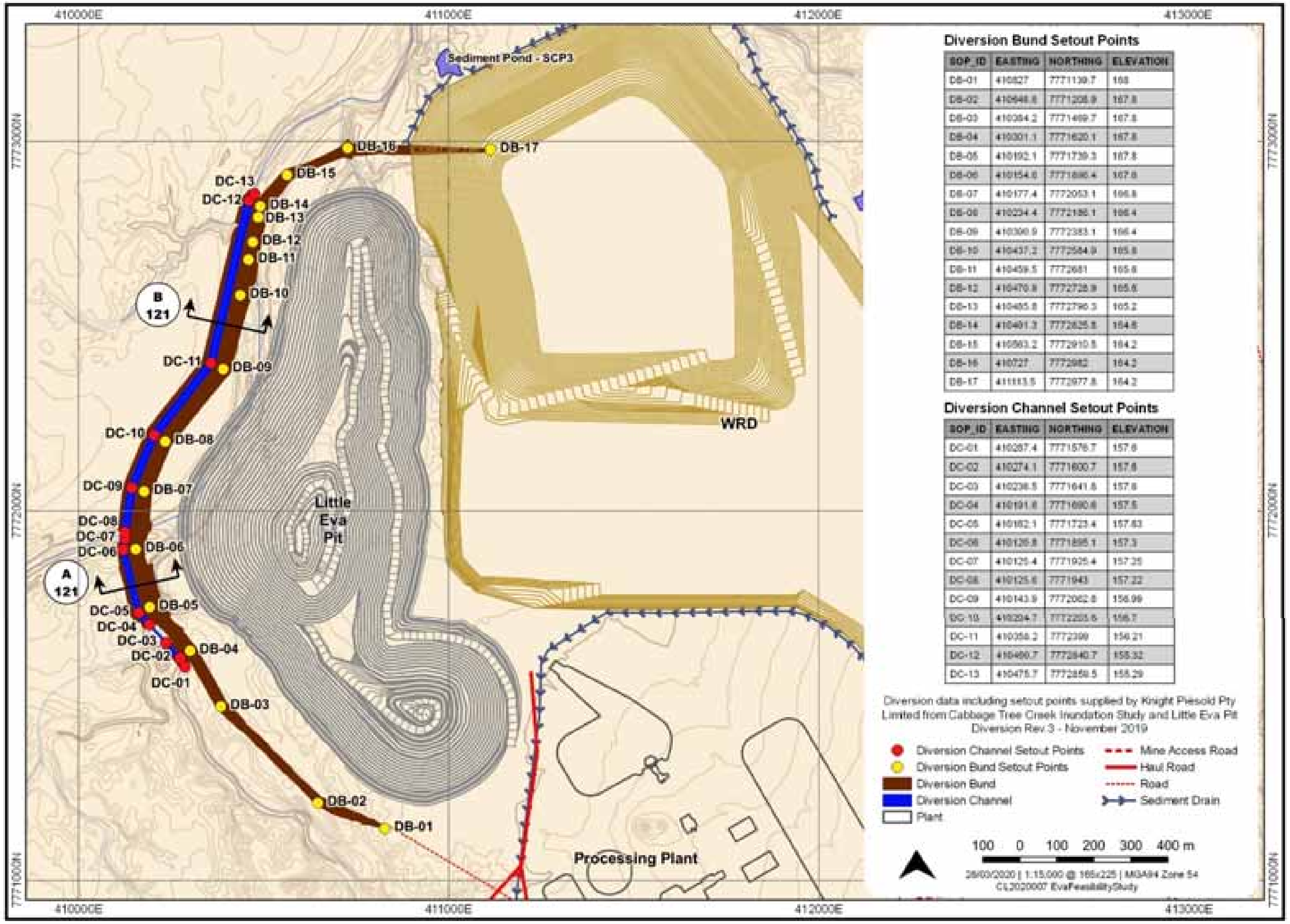


Figure 18-5: Cabbage Tree Creek Diversion Channel and Little Eva Pit Dike Plan

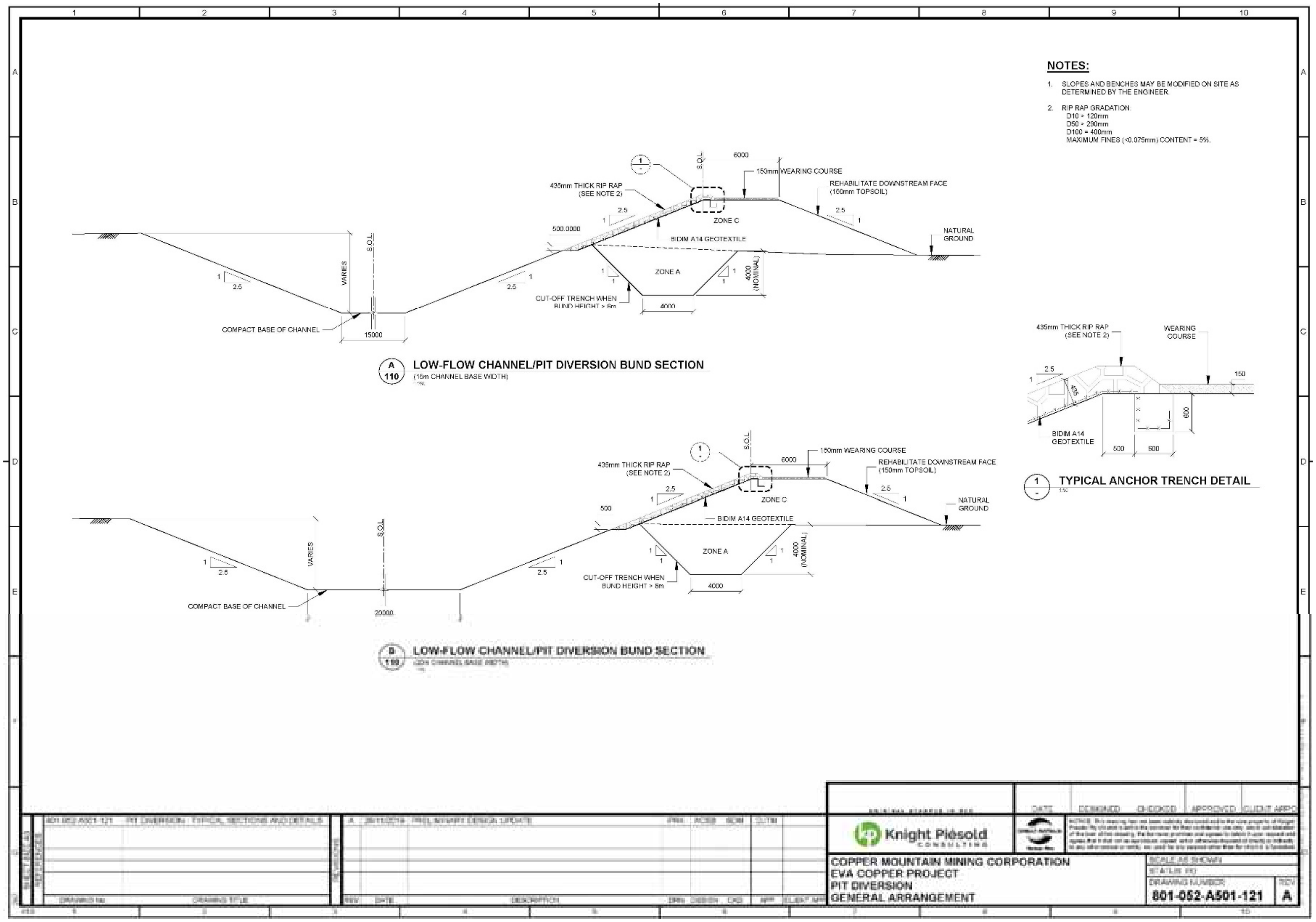


Figure 18-6: Cabbage Tree Creek Diversion Channel and Pit Bund Cross-Section Hydrogeology

18.4.4.1 Regional Hydrogeology

Hydrogeological investigations by Morgan and Associates (2007 and 2011) and Rockwater (2011 and 2018) have established that groundwater occurs in several aquifer systems.

18.4.4.2 Alluvial Aquifers

Unconfined alluvial aquifers are associated with deposits associated with contemporary drainage. The Project is located within the Cabbage Tree Creek catchment area and immediately adjacent to the main drainage channel. The alluvial aquifer is associated with extensive areas of colluvial outwash fans, paleochannels, and raised alluvial deposition. Groundwater occurs within the deeper parts of these alluvial deposits, and groundwater levels range from 5 m to 25 m below ground level. Well yields are high (airlift yields up to 18 L/sec) at the Cabbage Tree Creek prospect area, where the wells intersected calcretes at the base of the paleochannel, and water quality is generally fresh. The western pit wall extends across Cabbage Tree Creek and its linked, sediment-filled paleodrainage, which is likely to receive groundwater recharge from seasonal surface water flow within Cabbage Tree Creek.

18.4.4.3 Sedimentary Basin Aquifers

Deeper, fractured-rock aquifers have been located within the Landsborough Graben Phanerozoic sedimentary basin located between Cabbage Tree Creek in the east and Leichhardt River in the west. The key targets are mid-Cambrian limestone and sandstone units that are unconformably overlain by flat-lying Mesozoic sediments, comprising quartzose and sub-labile sandstone, siltstone, mudstone, and minor conglomerate. The limestone unit is truncated to the east by the Coolullah Fault, and to the west by the Pinnacle Fault. In this area the Mesozoic sediments are generally located above the standing groundwater level, and so do not form part of the Great Artesian Basin aquifer. Well yields are high (pump test sustainable yields of 10 L/sec to 15 L/sec) at the Cabbage Tree Creek prospect area where the wells intersected fractured rock aquifer immediately beneath the paleochannel; water quality is generally fresh to marginally brackish.

18.4.4.4 Crystalline Rock Aquifers

Aquifers occur in the regolith above crystalline Proterozoic rock in saprolite-saprock, in oxidation zones along fractures, and in vuggy, leached developments in carbonated rocks. The saprolite zone is linked to oxidation processes within the Little Eva mineralized copper-gold deposit, and copper-only deposits such as Blackard. Groundwater occurs at varying depths. In general, deeper zones of oxidation in the lowland area have groundwater at a depth 15 m to 25 m below the surface. Fractures extend into crystalline rocks below the general base level of pervasive oxidation to depths of 50 m to 120 m and, in places, form effective zones for water transmission. Carbonate-enriched rocks often show leached zones of vuggy development where they are crossed by faults and fractures. Principal water-transmitting locations in the crystalline rocks are closely related to zones of structural complexity involving faults and folds, particularly where they cross brittle rock contacts and carbonate rocks. At Little Eva, faults have produced wide brittle rock and shear structures. High water-transmitting zones occur in tension zones and vuggy formations associated with contacts with porphyry and carbonated units.

18.4.4.5 Groundwater Recharge

Recharge of the aquifers in the Project area is closely related to rainfall. This relationship involves both rainfall intensity that generates runoff ponding and local saturation of the soil profile, to prolonged rainfall events that result in regional saturation of the vadose zone and transmission to the aquifer through selective conductive pathways or megapores (Morgan, 2011). Three recharge pathways are evident:

- Direct intake from rain through a thin soil/scree cover, or directly into outcropping fractured crystalline rock, as found extensively in the upland areas or close to local escarpment faces around Knapdale Range. These fracture zone aquifers show rapid changes in water level and have high groundwater gradients leading to rapid drainage. The aquifers generally have low storage capacity.
- Transmission of rain water through thick soil, alluvium, saprolite, and saprock cover to fractured crystalline rock or sedimentary formations that host the regional aquifer system.
- Infiltration of creek surface water flow into the underlying, unconfined alluvial and paleochannel aquifers. The intensive, short duration storm events that are characteristic of the summer (December to February) season result in rapid generation of stream runoff from escarpment slopes to provide groundwater recharge of low salinity groundwater. This type of flow system may result in groundwater recharge during river flow periods, and groundwater discharge during the dry season periods. Creeks only flow for short durations (hours to weeks) following rain, after which river beds become dry or contain only isolated pools.

18.4.5 Groundwater Quality

Detailed analysis of groundwater samples from various drill holes at the Little Eva deposit found that groundwater is generally fresh to brackish, with total dissolved solids (TDS) ranging from 600 mg/L to 3,150 mg/L (Rockwater, 2011). These waters generally have close affinity with rainfall recharge and active solution of the carbonate-rich regolith. Strong metal anomalies (where present) appear to be restricted to known mineralized zones (Morgan, 2011).

Ionic ratios were examined by Morgan (2011) to provide water type identification as guidance to some processes involved in the aquifer system, including water sourcing, from precipitation to intake and through regolith processes, to the current condition of water in rivers and aquifers. Little Eva deposit drill holes were found to have similar ionic compositions. They were low in calcium relative to magnesium, and high in bicarbonate, suggesting dissolution of carbonate minerals. In most drill holes, the sulphate was low relative to bicarbonate, a feature of immature water. Sodium plus potassium levels were also higher than that of chloride, indicating dissolution of silicate or ion exchange within the regolith.

Groundwater in the unconfined alluvial and Cambrian sediments in the Cabbage Creek Borefield were fresh to marginally brackish, with salinities ranging from 330 mg/L to 850 mg/L TDS, dominated by calcium and bicarbonate ions, indicating a young age and good recharge conditions. The groundwater is slightly alkaline, with pH of 7.71 to 7.79. The Ryznar Stability Index indicates the water should not be corrosive or encrusting. The water has low metal and nitrogen concentrations, and is likely to be suitable for potable use.

18.4.6 Groundwater Usage

Groundwater is utilized by the low-intensity, open-range cattle grazing industry, via wells and drill holes sunk into shallow alluvial deposits. The only pastoral drill holes likely to be influenced by mine dewatering are Cabbage Tree Creek drill holes on the western side of the Little Eva pit. These water sources will be relocated in consultation with the landowner. Mining and dewatering will not impact the availability of groundwater to other users in the area. In addition, the proposed extraction of groundwater will not impact the recharge of the Great Artesian Basin or river flows subject to the Water Plan (Gulf) 2007 and associated moratorium. The extraction of groundwater at Little Eva will have a negligible impact on groundwater systems in the region.

18.4.7 Satellite Deposit Hydrogeology

No hydrogeological work has been carried out at the satellite deposits. Prior to mining, work will be carried out to assist with a detailed mine design. Blackard and Legend did have hydrogeological work carried out. Prior to mining additional work will be carried out to assess with a detail mine design.

18.4.8 Mine Dewatering and In-Pit Collection

18.4.8.1 Mine Dewatering

Mining at the Little Eva deposit will necessitate dewatering of the orebody aquifer. Studies by Morgan (2007 and 2011) have indicated that the principal water-bearing zone is the leached upper contact between the intrusive porphyry body and the overlying calc-silicate rock unit. Pit dewatering will be achieved by sustained pumping rates of 4,000 m^3/d. This would involve perimeter production wells Figure 18-7 and Figure 18-8), in-pit sumps, and sub-horizontal wall drain holes. The wall drain holes would target underdrainage of the broken domain identified in the northern portion of the western pit wall (please refer to Section 9 (Geotechnical) of this report, and note that this domain was previously incorrectly interpreted as the Cabbage Tree Creek/Coolullah Fault system), as well as a fault on the eastern side of the pit that converges with the Coolullah Fault to the north of the pit.

Once the main orebody aquifer has been dewatered, groundwater inflow to the pit will markedly reduce, and be confined to the lower parts of pit walls. Reduced groundwater flow conditions will occur because of the cone of depression extending into the less conductive rocks of the Corella Formation. Groundwater seepage (inflow) to the pit will be maintained by seasonal recharge from creek flow into the fracture systems below the creek beds bounding the sides of the pit.

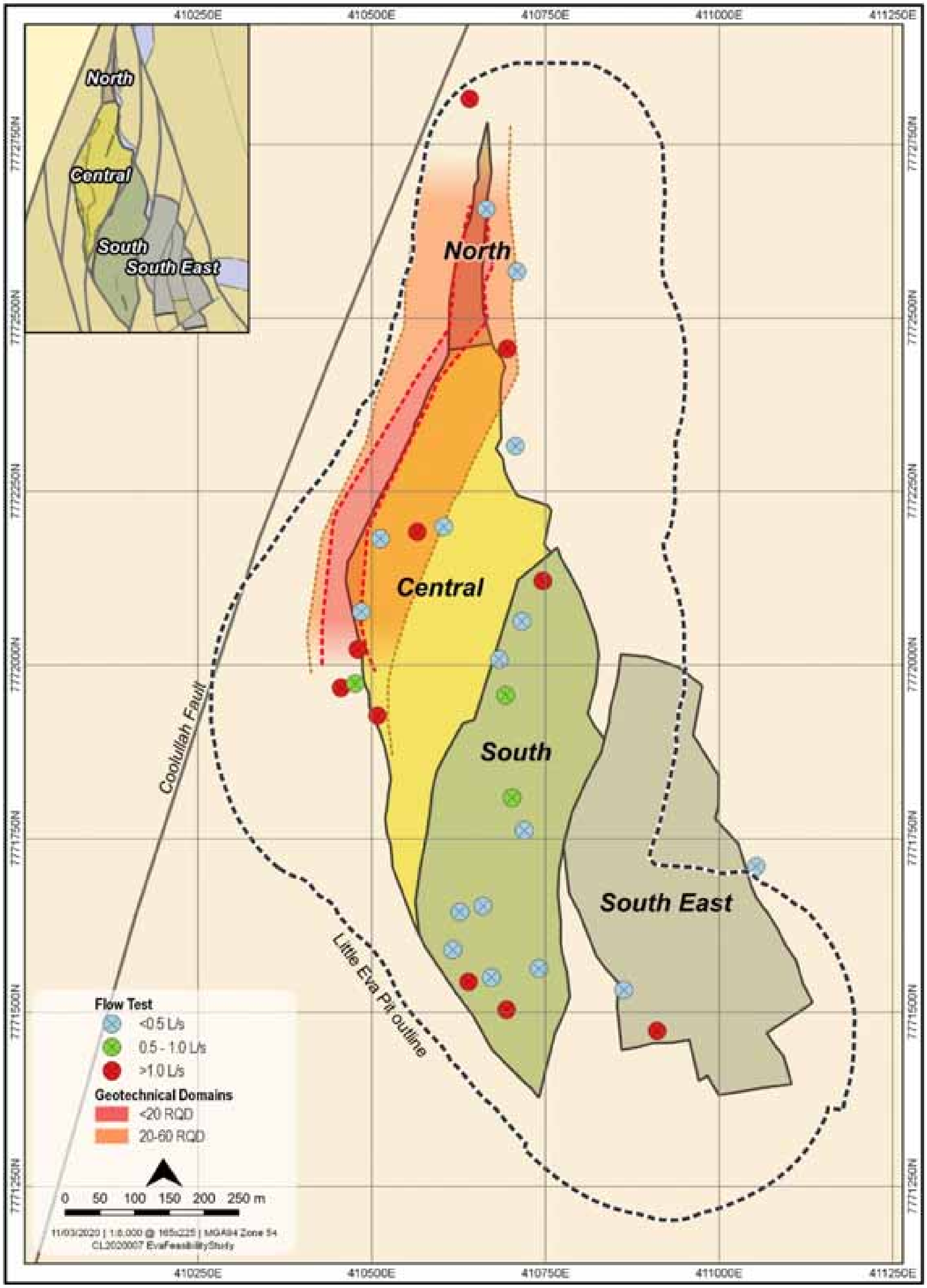
A cross-section showing the impact of dewatering during the Stage 1 and Final pits is provided in Figure 18-9. Ongoing dewatering activity to maintain dry mining conditions will result in a gradual increase in the lateral extent of the cone, rather than a deepening of the cone. The dewatering cone of depression footprint will retreat to approximately 500 m distance from the final Stage 6 pit crest (Morgan and Associates, 2012). An estimated drawdown at the time when equilibrium is established within the regional aquifer is presented in Figure 18-8.

Morgan and Associates (2012) revised the mine dewatering model for the Little Eva deposit and gave the following recommendations:

- When considering the mine dewatering strategy, potential pumping rates need to be set based on ore processing water supply needs, rather than just mine dewatering needs, as all dewatering discharge water will be used in the process water circuit. This will probably ensure that dewatering impacts stays well ahead of mining needs (dry mining conditions).

- The required pumping regime to achieve the modelled drawdown can be accomplished without development of drill holes LEPB03 and LEPB08. This will adequately meet dewatering requirements for the Stage 1 starter pit.
- At completion of Stage 1, the aquifer level will be drawn down close to its base at 160 m or approximately 0 m Australian Height Datum (AHD). This level is penetrated by Stage 2 through Stage 6 pit developments, and ongoing pit dewatering will be achieved largely through pit wall drains. Little groundwater is expected to seep from below the 0 m AHD level. It is therefore important that adequate wall drains are established prior to mining below the 0 m AHD level, as behind-wall water sources will no longer be accessible to a drill rig located in the pit as the mine gets deeper.
- Additional dewatering wells, and/or a series of 100 m to 150 m deep, easterly-directed horizontal drill holes, will be required to adequately reduce groundwater gradients behind the southeast pit wall. The wells will need to be drilled from the haul ramp between depths of 100 m to 20 m AHD.

High flow rates in test water wells largely correspond to the Broken and Moderately Broken geotechnical domains on the western margin of the deposit (Figure 18-7). These contiguous domains were identified subsequent to the preliminary well field design in Figure 18-8. Opportunity exists to improve the borefield design by perimeter production wells targeting these zones.



Note: High flow rates largely correspond to the Broken and Moderately Broken domains on the western margin of the deposit.

Figure 18-7: Test Water Well Flow Rates vs. Deposit Geotechnical Domains

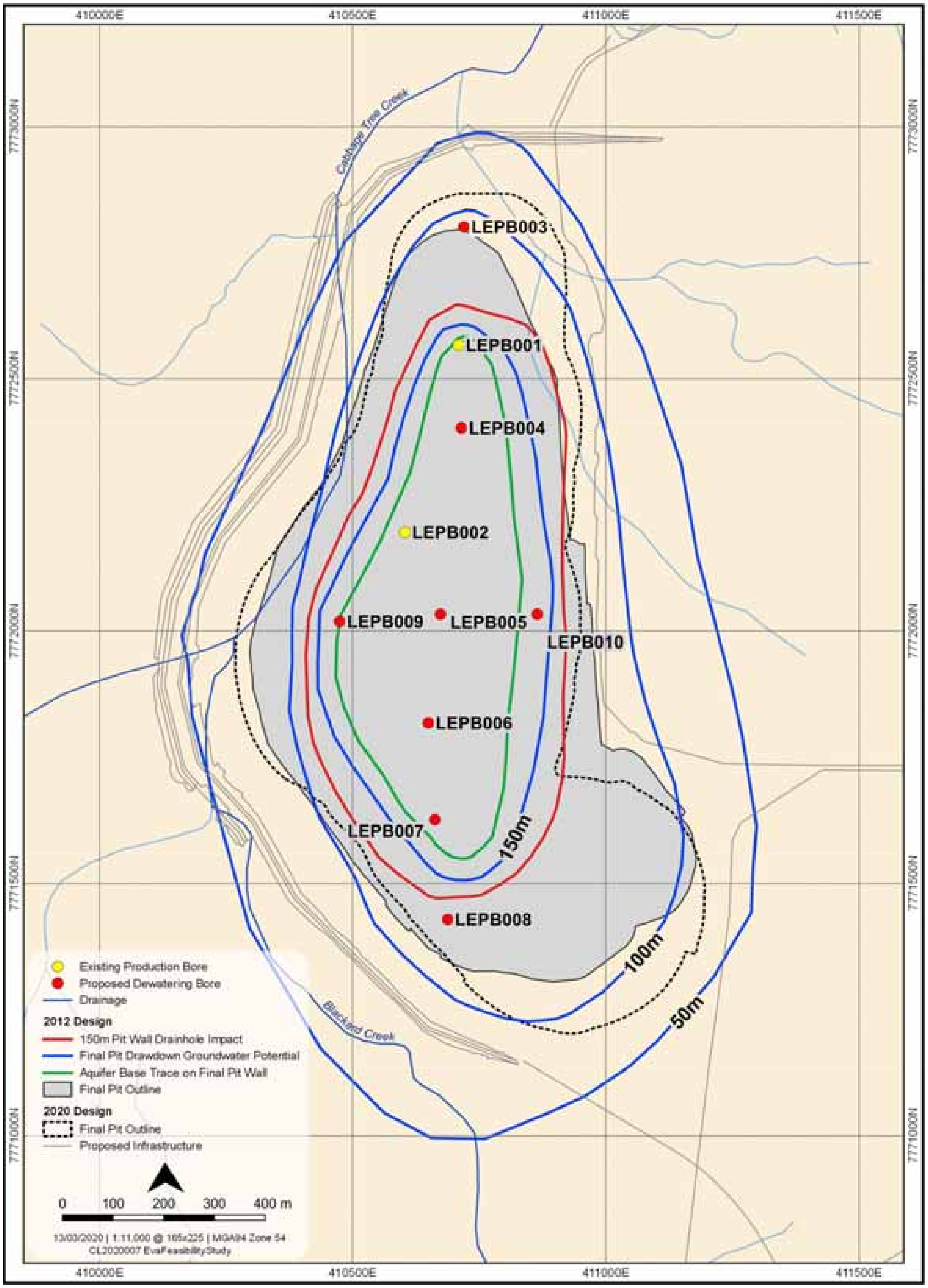


Figure 18-8: Groundwater Isopotentials, Little Eva Pit, and Proposed Water Wells

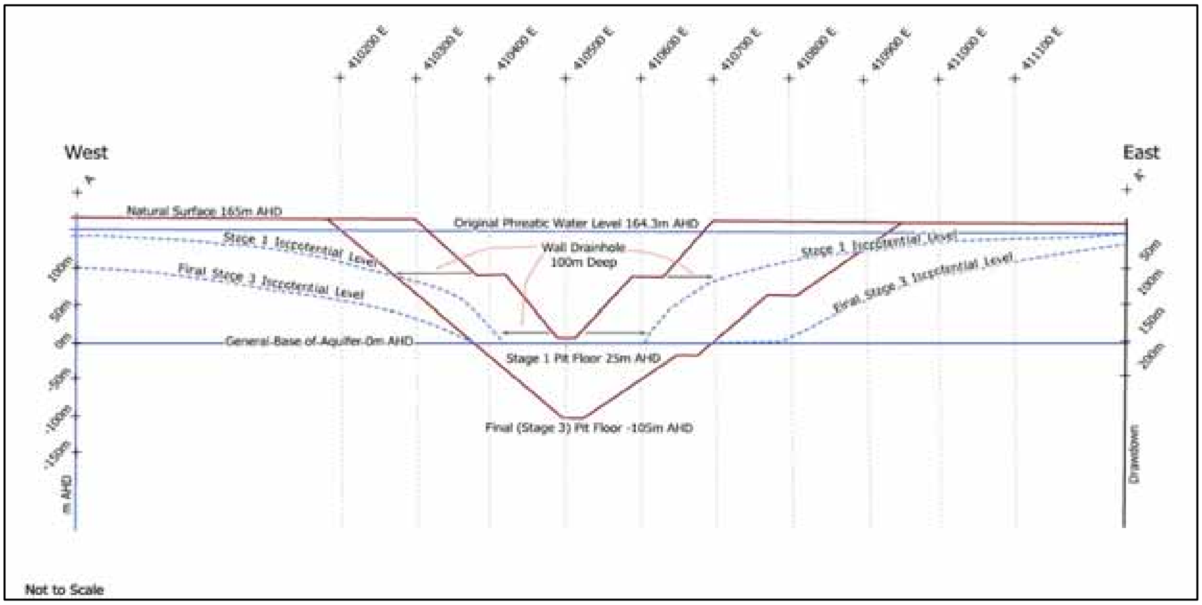


Figure 18-9: Little Eva Pit Conceptual Dewatering Configuration

18.4.9 Additional Mine Water Supply

As dewatering proceeds, discharge volumes will reduce, and it may become necessary to draw raw water for the process water circuit from other sources. Rockwater (2011) conducted an RC drilling exploration program to assess the water supply needs. A simple MODFLOW-based groundwater model was run to provide a first-order estimate of the potential volumes of water that could be produced by dewatering the proposed Little Eva pit. The model indicated that dewatering at a rate of 4,000 m^3/d (approximately 50 L/sec) over two years would produce a maximum drawdown of approximately 150 m. While the final Little Eva pit depth is planned at 250 m, Rockwater advised that 4,000 m^3/d of extraction be regarded as an upper limit, due to:

- Decline in permeability being a likely result of open-fracturing of the orebody, decreasing with depth
- Dewatering rate being based on production wells LEPB01 and LEPB02, which are particularly permeable, and possibly not representative of the entire Little Eva deposit
- Low permeability of the Coolullah Fault and some of the Cambrian sediments to the west is likely to form a barrier to the extent of drawdown and groundwater flow
- Yields from fractured rock aquifers can decline rapidly and unpredictably during dewatering, and therefore some redundancy in the water supply capacity is recommended.

Rockwater (2011) also suggested that further investigation should take place of other groundwater supplies, including:

- Cambrian and Mesozoic sediments of the Landsborough Graben
- Modern alluvial sediments associated with the channel and floodplain of Cabbage Tree Creek
- Cabbage Tree Creek copper target, approximately 3 km north of Little Eva.

In 2018, Cambrian and younger alluvial sediments west of the Coolullah Fault were subject of a water exploration drilling program (Rockwater, 2018). Three wells were established and test-pumped with substantial groundwater yields encountered in two. These wells will form the initial stage of the Cabbage Tree Creek borefield (Figure 18-13).

During 2019, a Passive Seismic survey was completed over part of the Landsborough Graben, west of Little Eva. The output of this survey was a model for depth to top of Cambrian strata which was used to guide drilling of two shallow vertical RC drill holes in the Western Depression area, approximately 6 km west of the proposed Cabbage Tree bore field. Both holes encountered water associated with an ~18 m interval of coarse sands and gravels from 28 m depth.

Studies by Morgan (2007 and 2011) on the impact of pit dewatering on groundwater sources have indicated that resources will require dewatering to depths of 150 m to 200 m below natural standing water level ranging from 5 m to 20 m below ground level. Hydraulic data indicates that the general Project area is composed of a rock sequence with low hydraulic conductive properties and, as a result, the hydraulic surface will rise steeply behind the wall toes of the pit floor and will have significant impact on natural water level at distances beyond 500 m from the periphery of each pit. This dewatering will have minimal impact on shallow water resources in adjacent creek and river channels because of high natural recharge rate during creek flow periods and low hydraulic conductive properties of the country rock at a shallow depth below the riverbeds. This section of rock is unlikely to fully drain towards the pits between recharge periods. The extent of the spread of dewatering on the country surrounding Little Eva, Blackard, and Scanlan pits is presented on Figure 18-10 to Figure 18-12.

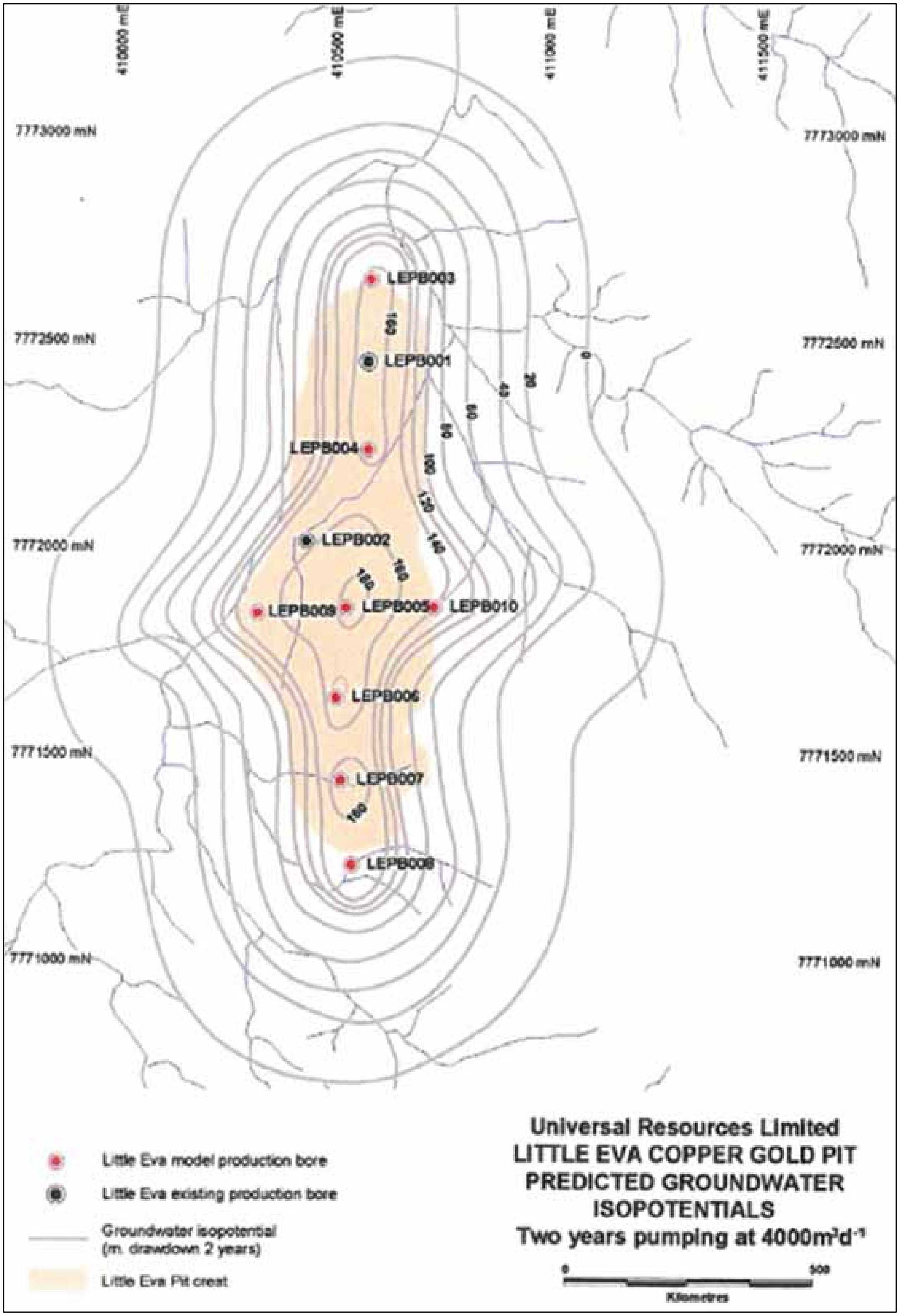


Figure 18-10: Little Eva Conceptual Pit Dewatering and Wells

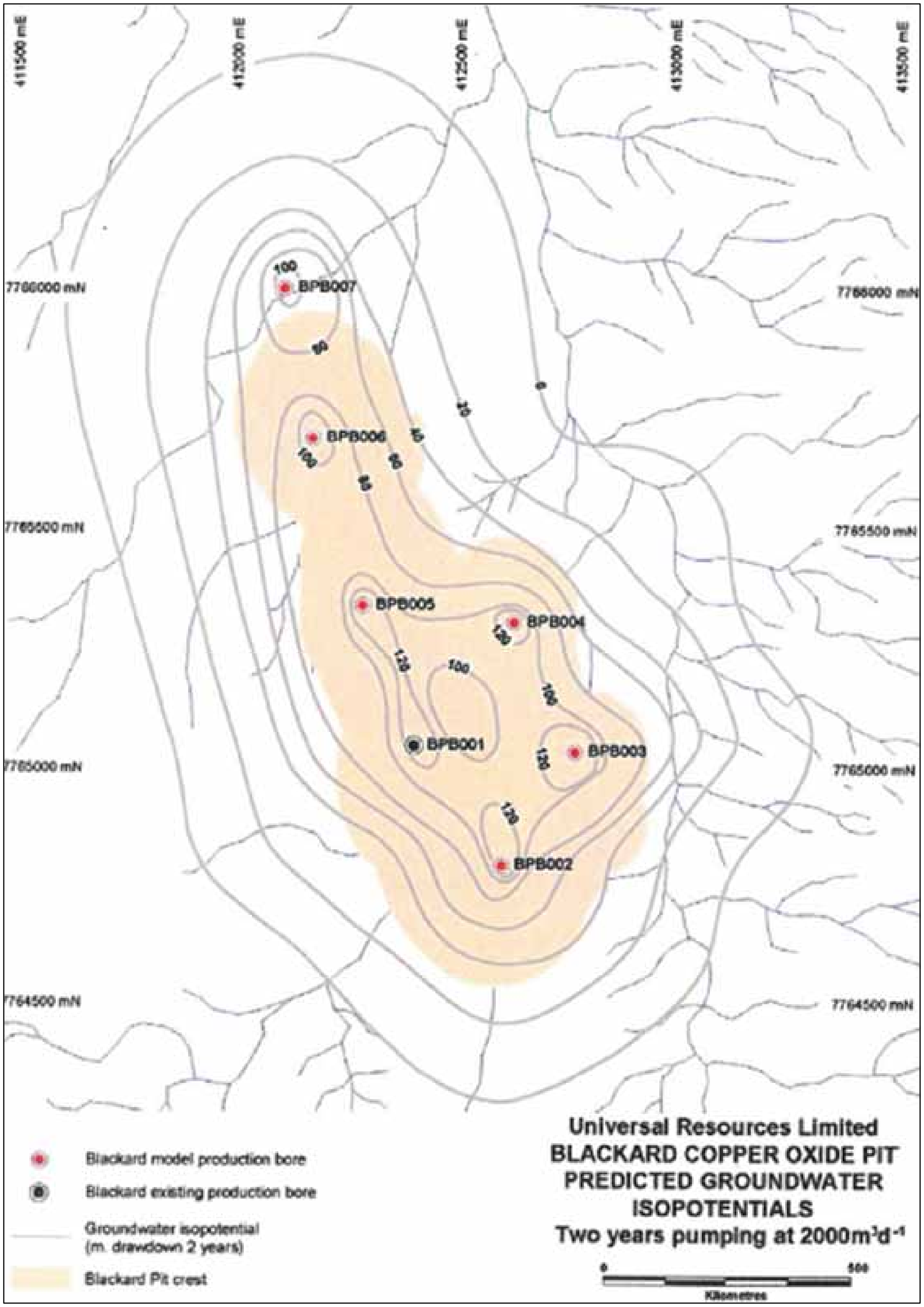


Figure 18-11: Blackard Conceptual Pit Dewatering and Wells

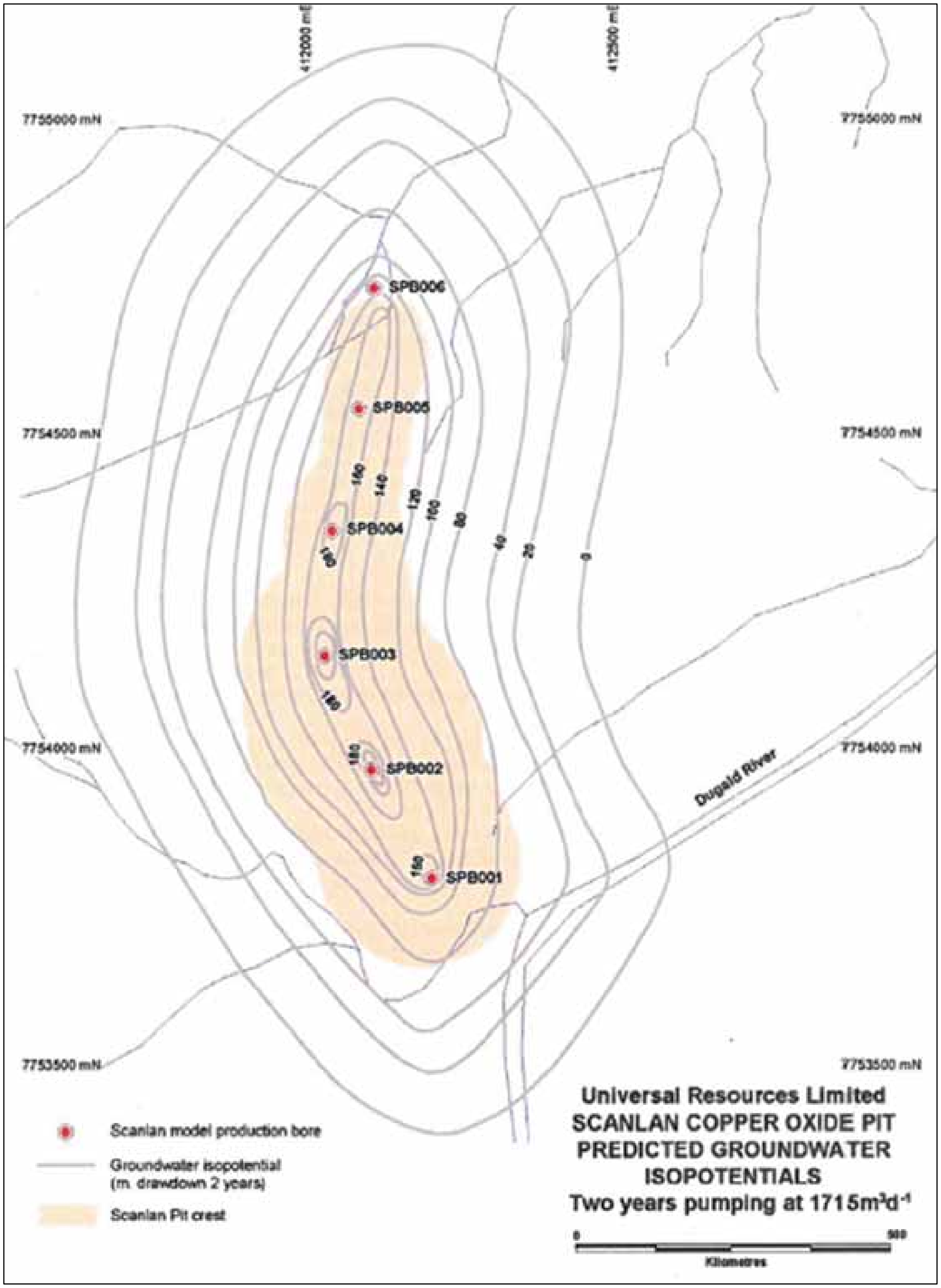


Figure 18-12: Scanlan Conceptual Pit Dewatering and Wells

18.4.10 Project Water Balance

KCB developed a high-level water balance for the Project based on the following assumptions:

- The water balance is an average annual water balance.
- Site demands have been developed by Copper Mountain Mining Corp. (CMMC).
- Pit groundwater dewatering rates:
  - KH Morgan and Associates (Morgan) (2009) estimates used for Little Eva, Blackard, and Scanlan pits (refer to Figure 18-10 to Figure 18-12)
  - Pit dewatering commences one year prior to the start of mining at the pit (i.e., Little Eva Pit – Year 0, Blackard and Turkey pits – Year 1, and Scanlan Pit – Year 5)
  - Yields were estimated to peak at approximately 3.0 GL/a, before tapering off to 2.4 GL/a by Year 9 as recharge rates are insufficient to replace pumped volumes.
- Borefield water supply:
  - The water supply is based on the design by Rockwater [2018], with an approximate 3 GL/a supply for five years, assuming 15 bores supplying ≅6.5 L/sec (refer to Figure 18-13)
  - The water balance is based on continued supply from the borefield over the LOM, or that a new borefield to make up the difference is identified in later years.
- Rainfall and runoff harvesting are assumed for the Little Eva and Blackard Pits only, and has been assessed using:
  - Wet conditions: 90th percentile annual runoff volume
  - Average conditions: 50th percentile annual runoff volume
  - Dry conditions: 10th percentile annual runoff volumes.
- As satellite pits are developed, these will provide additional water harvesting potential; water harvested from these additional pits has not been considered in the water balance.
- Water harvesting from the sediment ponds has not been included, as the supply is of limited reliability because of the proposed water use hierarchy (i.e., when water from the sediment ponds is needed, it may not be available due to evaporation loss).
- The hierarchy of water supply is:
  - Rainfall-and runoff captured in the mining pit
  - TSF decant return
  - Groundwater from pit dewatering
  - Borefield water supply.

The process plant will continuously discharge tailings at a slurry density of 63% solids to the TSF; however, return water volumes will be highly variable and dependent upon weather conditions. KCB high-level modelling indicates that during dry years (i.e., the year that corresponds to the 10th percentile lowest rainfall total), return water rates could be as low as 13%, and in wet years (90th percentile rainfall years), return water rates could be as high as 80%.

The high-level water balance is presented in Table 18-1.

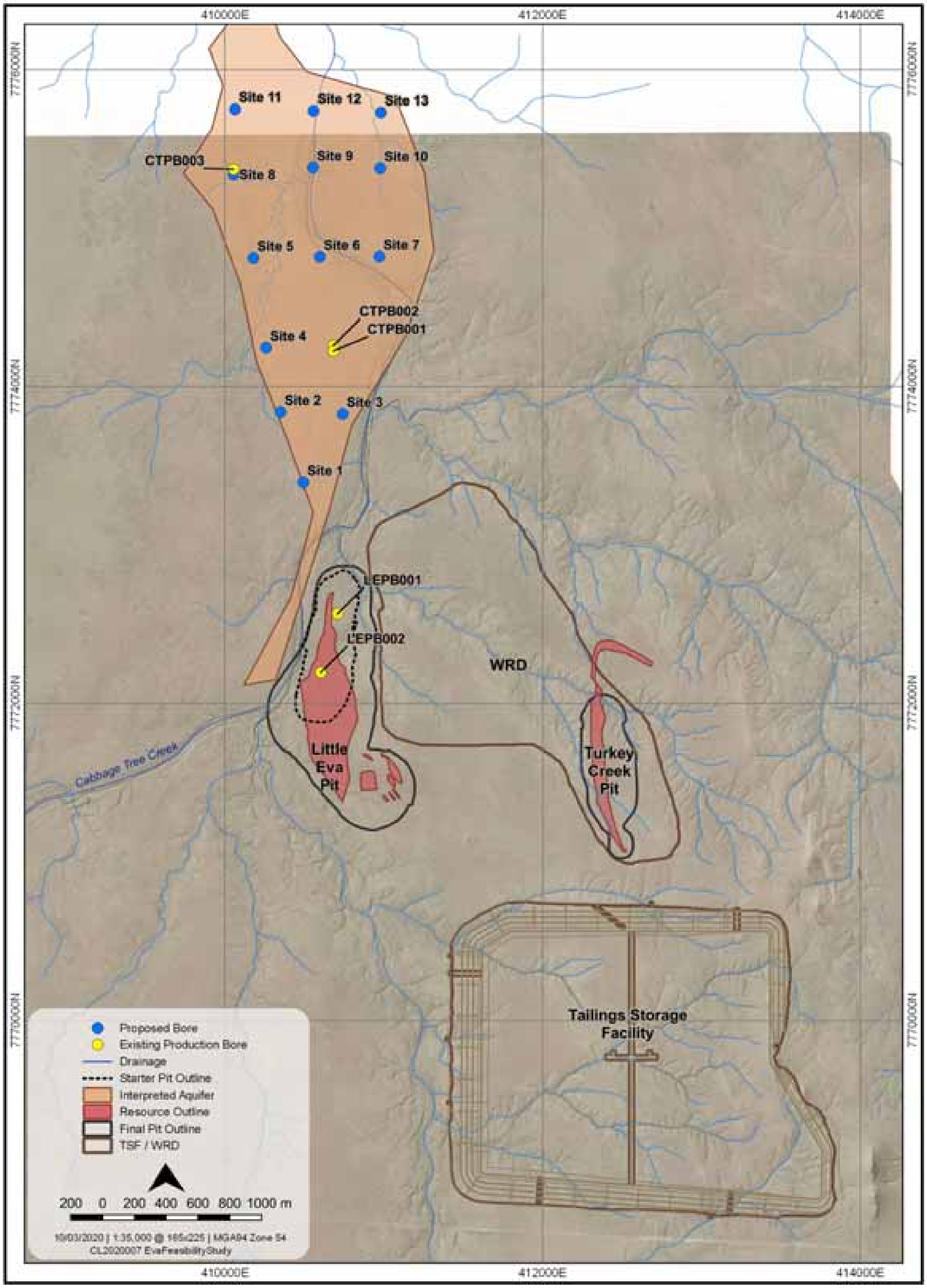


Figure 18-13: Cabbage Tree Creek Conceptual Borefield and Existing Wells

Table 18-1: Eva Copper Project Water Balance

Item	Sub-Items	Unit	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Notes
Site Water Demands																			
Mill	ROM	kt/a (wet)	0	11,740	11,740	11,772	11,740	11,740	11,740	11,772	11,740	11,740	11,740	11,772	11,740	11,740	11,740	10,860	
		kt/a (dry)	0	11,388	11,388	11,419	11,388	11,388	11,388	11,419	11,388	11,388	11,388	11,419	11,388	11,388	11,388	10,860	[1]
		%	3.0%	3.0%	3.0%	3.0%	3.0%	3.0%	3.0%	3.0%	3.0%	3.0%	3.0%	3.0%	3.0%	3.0%	3.0%	3.0%	[1]
		ML/a	0	352	352	353	352	352	352	353	352	352	352	353	352	352	352	336	
	Product	kt/a (wet)	0	517	517	551	517	517	517	551	517	517	517	551	517	517	517	498	
		kt/a (dry)	0	472	472	503	472	472	472	503	472	472	472	503	472	472	472	454	
		%	9%	9%	9%	9%	9%	9%	9%	9%	9%	9%	9%	9%	9%	9%	9%	9%	[1]
		ML/a	0	45.4	45.415	45.5	45.4	45.4	45.4	45.5	45.4	45.4	45.4	45.5	45.4	45.4	45.4	43.7	
	Tailings	kt/a (wet)		17,327	17,327	17,375	17,327	17,327	17,327	17,375	17,327	17,327	17,327	17,375	17,327	17,327	17,327	16,518	
		kt/a (dry)	0	10,916	10,916	10,946	10,916	10,916	10,916	10,946	10,916	10,916	10,916	10,946	10,916	10,916	10,916	10,406	[1]
		%	63%	63%	63%	63%	63%	63%	63%	63%	63%	63%	63%	63%	63%	63%	63%	63%	[1]
		ML/a	0	6,411	6,411	6,429	6,411	6,411	6,411	6,429	6,411	6,411	6,411	6,429	6,411	6,411	6,411	6,112	
	Net water demand	GL/a	0.00	6.10	6.10	6.12	6.10	6.10	6.10	6.12	6.10	6.10	6.10	6.12	6.10	6.10	6.10	5.82	
Haul Road	Dust suppression	GL/a	0.25	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	[1]
Crusher/ stockpile	Dust suppression	ML/a	0.00	470	470	471	470	470	470	471	470	470	470	471	470	470	470	434	[1]
Other	Admin/ truckshop/ washpad	ML/a	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	[1]
Total Site Demand		**GL/a**	**0.29**	**7.11**	**7.11**	**7.13**	**7.11**	**7.11**	**7.11**	**7.13**	**7.11**	**7.11**	**7.11**	**7.13**	**7.11**	**7.11**	**7.11**	**6.79**	
Water Supply																			
Borefield	Total supply	GL/a	3.07	3.07	3.07	3.07	3.07	3.07	3.07	3.07	3.07	3.07	3.07	3.07	3.07	3.07	3.07	3.07	[2]
Pit dewatering	Groundwater extraction	GL/a	1.46	2.92	2.92	2.41	2.41	2.56	2.56	2.48	2.48	2.37	2.37	2.37	2.37	2.37	2.37	2.37	[3]
TSF return	Wet conditions	GL/a	0.00	5.13	5.13	5.13	5.13	5.13	5.13	5.13	5.13	5.13	5.13	5.13	5.13	5.13	5.13	5.13	[4]
	Average conditions	GL/a	0.00	1.73	1.73	1.73	1.73	1.73	1.73	1.73	1.73	1.73	1.73	1.73	1.73	1.73	1.73	1.73	[4]
	Dry conditions	GL/a	0.00	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	[4]

Item	Sub-Items	Unit	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Notes
Total Inflow	**Wet conditions**	**GL/a**	**4.53**	**11.12**	**11.44**	**10.94**	**10.93**	**11.07**	**11.23**	**11.17**	**11.16**	**11.05**	**11.05**	**11.06**	**11.05**	**11.05**	**11.05**	**10.81**	
	Average conditions	**GL/a**	**4.53**	**7.73**	**7.82**	**7.32**	**7.31**	**7.46**	**7.51**	**7.44**	**7.43**	**7.32**	**7.32**	**7.33**	**7.32**	**7.32**	**7.32**	**7.24**	
	Dry conditions	**GL/a**	**4.53**	**6.83**	**6.85**	**6.34**	**6.33**	**6.48**	**6.49**	**6.42**	**6.42**	**6.31**	**6.31**	**6.31**	**6.31**	**6.31**	**6.31**	**6.27**	
Water Balance (+ve = surplus, -ve = shortfall)																			
Wet conditions		GL/a	1.18	0.94	1.25	0.74	0.74	0.89	1.05	0.97	0.97	0.86	0.86	0.86	0.86	0.86	0.86	0.94	[5]
Average conditions		GL/a	1.18	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	[5]
Dry conditions		GL/a	1.18	-0.28	-0.26	-0.79	-0.77	-0.63	-0.62	-0.71	-0.69	-0.80	-0.80	-0.82	-0.80	-0.80	-0.80	-0.52	[5]

Notes: [1] Sources: CMMC [EvaCopper_R12_working_file(release3-AnnualizedScheduleWithPits).xlsx, 10435601-MB-0001_C.xlsx, or NI 43-101 Technical Report for the Eve Copper Project Feasibility Study]
[2] Source: Rockwater, 2018
[3] Source: Morgan, 2009
[4] Calculated: KCB, includes Eva Pit rainfall capture in the pit is pumped to the TSF
[5] When the pit dewatering TSF return can supply the site water demands, the bore field supply is reduced for the water balance calculation.

18.4.10.1 Impact of Extreme Conditions on Water Balance

In dry conditions, the Project may require additional makeup water from alternate sources.

Conversely, during wet periods, there would be rainfall and runoff captured in the pit that will be pumped into the TSF and then used to supply site demands. The Project is not seeking to apply for a licence for release of mine water into the regional watercourses: during wet periods, water will be transferred to the TSF, where it will be a water source for the process mill.

18.4.11 Discharge to Environment

Based upon modelling of average climatic conditions, the borefield supply can be managed to keep the operation water neutral, thus it is not anticipated that there will be any requirement to discharge groundwater to the environment. However, prior to plant start up, dewatering of the Little Eva pit Stage 1 will need to begin as pre-strip is carried out and initial stockpile material is produced. It is expected that this water will be used for construction activities and haul road dust suppression. During this time, pit water drawdown rates will be matched to rates of vertical mining advance, avoiding the generation of surplus water. In the event that drawdown rates necessarily exceed water usage, the excess will be stored in site raw and process water dams that will be constructed early.

18.4.12 Additional Water Supply

No investigation has been carried out on groundwater yields associated with dewatering the satellite deposits, except for Blackard and Scanlan. As a result, the Blackard pit dewatering has been considered in the water balance.

Further water supply investigation will be undertaken in the early years of the Project to provide adequate make-up water from other sources as yield drops from the existing sources. There may be potential to source additional water from:

- Construction of further wells in the Cabbage Tree Creek borefield area
- Construction of wells in other areas of the Landsborough Graben
- Harvesting surface water: conditional upon approval by the environmental authorities, surface water could be harvested during the wet season and used for processing, allowing the established water wells to recharge, extending their life, and providing greater flow rates during the dry season
- Allocation from the Lake Julius to Ernest Henry pipeline owned by SunWater. The capital cost for this option is not included in the estimate, Section 21.

18.4.13 Construction Water

The Cabbage Tree Creek test wells (CTPB02 and CTPB03) will be expanded to supply raw water for construction into the early works supplied fresh water/firewater tank, which is 14.8 m in diameter, with a height of 9.2 m (1,575 m^3). The wells will be capable of providing adequate water for construction. The accommodation village will have a dedicated well developed, as indicated earlier.

18.4.14 Environmental Authority Considerations in Relation to Water

Universal generated an Environmental Impact Statement (EIS), which was submitted to the Queensland Environmental Protection Agency (QEPA) in 2007. QEPA subsequently issued an assessment on the EIS in 2008. The EIS contemplated the potential impacts of the Project on regional groundwater and surface water. There has been significant change to the Project parameters since that time; however, some of the information remains relevant to the current Project.

To facilitate the granting of an Environmental Assessment for the Project in 2011, Altona generated an Environmental Management Plan (EMP), which was submitted to Queensland's Department of Environment and Heritage Protection (DEHP). A key part of the EMP outlined the potential impacts the Project would have on groundwater and surface water, and proposed a plan for its monitoring and management. In July 2012, the EA was granted. Subsequently, the Project parameters changed, with the expansion of the Little Eva pit, the inclusion of more satellite pits, removal of Scanlan and Blackard from the mine plan, a higher throughput process plant, and a larger diversion of Cabbage Tree Creek around the west side of the Little Eva pit. As the changes to the Project parameters were considered significant, a revised submission was provided to Queensland's DEHP to facilitate the granting of an EA amendment. The EA covering the 2016 Project Layout (that did not include the Cabbage Tree Creek Borefield) was submitted in March 2016, and approved in July 2016.

In determining impact and management of the water resources, the submissions all considered the following key aspects:

- Ephemeral surface water in local watercourses is potentially a source of water for stock animals, and it is of environmental value to the local landholders and users downstream of the Project.
- The watercourses can also be considered to have environmental value for aquatic ecosystems adapted to the ephemeral systems of North West Queensland.
- The groundwater quality is of environmental value to local landholders, and therefore protection of existing groundwater quality must be maintained to ensure ongoing and sustainable use of this resource.
- The potential groundwater water quality impacts from Project activities could include infiltration to groundwater of process water, mine water, sewage water, or leachate containing elevated concentrations of Total Dissolved Solids (TDS), sulphate, metals, or low pH water.
- Groundwater from wells and drill holes sunk into shallow alluvial deposits on creeks is used by the low-intensity open range cattle grazing industry.

Water assessments have determined that mining and dewatering will not impact availability of this resource. The proposed extraction of groundwater associated with the Project will not affect the recharge of the Great Artesian Basin or river flows that were subject to the Gulf Water Resource Plan and associated moratoria. The extraction of groundwater will have a negligible impact on groundwater systems in the region.

The regional site layout incorporates water and sediment management structures. Any surface runoff within process areas is deemed to be contaminated, and as a result, this water needs to be contained and collected for reuse in the process plant. The TSF design includes water quality monitoring wells for ongoing assessment of seepage that may be transmitted through the liner.

Concentrate transport trucks will be subject to wheel wash and sediment collection prior to exit from the concentrate loading area, with the water recycled to the process plant for reuse. Sediments from the wheel wash will be periodically removed and returned to the process plant. Drainage from the concentrate shed and handling area will be directed to a containment pond, where water will be captured and returned to the processing water system for reuse.

Water resource quality baseline information has been established by the Project via annual reports of water quality and sediment monitoring submitted to the Queensland authorities over multiple wet and dry seasons.

Ongoing Surface and Groundwater Monitoring Plans that include surface sampling and monitor wells, along with associated trigger values, have been included in the current approved EA.

18.4.15 Water Extraction License Considerations

The licensing requirements for the two borefield locations at the Little Eva and Blackard pits included in the approved EA were reviewed by the Water Management section of the Department of Natural Resources, Mines and Energy (DNRME) in January 2017. Background on the location of the geological units that water will be extracted from was provided.

The assessment concluded that, while the two borefield locations fall within the footprint of the Carpentaria Management Area of the Water Plan (Great Artesian Basin) (GAB WP 2006), the DNRME indicated that it was satisfied that the borefields are unlikely to directly take GAB water from the GAB management units (geological formations), and while there is limited data to determine if water will be taken indirectly, the risk to the basin and other users is low.

Based on this assessment, the current advice received in 2017 and again in 2018 was the following:

- A water license or associated water license (AWL) will not be required for dewatering operations at the Little Eva pit.
- A borefield tapping non-GAB sediment will not require a water license. Such is the case for the proposed Blackard Borefield, which taps aquifers within either weathered/oxidized saprolite zones, or fracture zones in crystalline basement rocks that are much older than the management units. The borefield at Cabbage Tree Creek, which was established after the last EA amendment, has not yet been addressed; however, these wells similarly do not intersect GAB management units.

The Project will be subject to Chapter 3 of the *Water Act* 2000; however, it will be exempt from the need to prepare Underground Water Impact Reports (UWIR) and Baseline Assessment Plans (BAPs).

18.5 Tailings Storage Facility

18.5.1 General

Paterson & Cooke (P&C) was retained to conduct a conceptual study comparing thickened slurry, versus paste and drystack tailings. Refer to the report Eva Tailings Dewatering Conceptual Study, Paterson & Cooke, November 2019 P. The scope included the following options:

- Option 1: Dewatering tailings to thickened tailings (about 60% mass solids concentration) and pumping to the tailings storage facility

- Option 2: Dewatering tailings to thickened paste tailings (about 69% mass solids concentration) and pumping to the tailings storage facility
- Option 3: Dewatering tailings to filtered tailings (about 85.5% mass solids concentration) and trucking to the tailings storage facility.

Table 18-2 presents the conclusions of the P&C study.

Table 18-2: Evaluation Outcome of the Eva Tailings Dewatering

Option	Capital Cost (US$)	Operating Cost (US$/t)
1	23.6	0.39
2	65.2	0.63
3	111.1	2.10

Based on the above conclusion CMMC and KCB selected the thickened tailings option for this study.

The Company completed a DFS for the Project in July 2012, which had the TSF located over the Turkey Creek deposit. Knight Piésold was engaged to perform geotechnical investigations and site selection studies for the new TSF location, directly south of the processing plant. Knight Piésold completed a feasibility study for the relocated TSF in 2019. KCB then updated the Knight Piésold design for an increased tailings storage requirement, increasing the design capacity of the TSF from 91.3 Mt to 170 Mt of tailings.

The Eva TSF will be a two-cell paddock facility designed to contain 170 Mt of tailings over approximately 15 years of mine life. The East and West Cells can be operated independently and are separated by a rockfill centre wall, positioned to create cells of approximately equal area. At the ultimate embankment height (maximum ~52 m), the West Cell will have a total footprint area of 216 ha, and the East Cell 208 ha, for a total disturbed footprint area of 424 ha for the TSF.

The TSF will be constructed using mine waste, primarily delivered and placed by the Company's mining fleet. An upstream low permeability zone on the embankments and a low permeability basin soil liner will be placed by a civil contractor, under the supervision of the geotechnical engineer. Supernatant water will be collected and pumped from decant towers for reuse in the process plant. The embankments will include internal drainage to reduce the phreatic surface to increase stability.

In the event of a dam break, the Eva TSF has been assessed to be a High consequence structure according to the Department of Environment and Science (DES) guidelines. The TSF will be a regulated structure. In the event of failure to contain overtopping, the consequence category is Significant. Under ANCOLD (2019) guidelines, the Eva TSF has a consequence category of High A.

The TSF has been designed with enough storage capacity to contain runoff from seasonal rainfall events without unauthorized release of contaminants. The TSF has a Significant consequence category for the failure to contain overtopping scenario, and accordingly requires a dam storage allowance (DSA) volume corresponding to a 1-in-20 annual exceedance probability (AEP), two-month duration wet season volume, plus expected process inputs during the critical wet season period. The emergency spillways for the East and West cells have been sized for the probable maximum flood (PMF), with no additional freeboard.

Seepage and stability analyses have been completed for the LOM and intermediate configurations of the facility. Stability analyses indicate that under static, seismic, and long-term conditions, the TSF meets ANCOLD (2019) design criteria consistent with a High A consequence category facility. This design is currently in the feasibility stage and will require additional field investigations and studies for detailed design and prior to construction of the starter embankments.

The performance of the TSF will be monitored through a series of vibrating wire piezometers, groundwater quality monitoring bores, and regular ground surveys. The TSF closure plan is to rehabilitate the facility progressively to develop a self-sustaining landform that supports vegetation (i.e., a dry cover) and end-use land goals, which will be defined in the site's Progressive Rehabilitation and Closure (PRC) plan.

18.5.2 Tailings Management

Tailings will be discharged into the facility by subaerial deposition methods, using a combination of spigots located at regularly spaced intervals along the perimeter embankments, as well as the first 300 m of the rockfill centre wall. Supernatant water will be removed from the TSF via submersible pumps located within decant towers. Three decant towers will be needed over the lifetime of the facility. The Stage 1 decant tower will be located at the centre of the western wall perimeter wall, at the local low point of the natural topography. To account for the pond being close to the perimeter walls during initial operation, the Stage 1, 2, and 3 embankments have been designed with protective filters downstream of the core to prevent internal erosion and mitigate the impacts should continuous cracks penetrate the low-permeability zone of the embankment

Beyond Stage 1, the design intent is to shift the pond away from the perimeter embankments to the central dividing wall between the two cells. This is where the East and West decant towers will be located. The pond location will be controlled by targeted deposition and regular rotation of the active tailings beach. Solution recovered from the decant system will be pumped back to the processing plant site for reuse in the process circuits.

The TSF has been designed to be constructed in stages. The West Cell will be constructed first as the starter embankment. The East Cell will be built from Stage 2 onwards and will use cut roads through the natural topography for tailings deposition from the east. From Stage 7 onwards, the tailings level will cover the natural topography and the facility will become a "turkey's nest" style impoundment (i.e., continuous perimeter embankments). The embankments will be raised using downstream construction methods for the initial three stages, followed by centreline construction methods for the remainder of the operation.

To reduce seepage from the TSF, a low-permeability soil liner will be constructed over the entire TSF basin area. This liner comprises a compacted 300 mm layer of low-permeability soil, either reworked in-situ material or imported Zone A material. The low-permeability liner will be covered with a 150 mm protection layer, comprising the existing basin surficial material or mine waste to reduce erosion during operation prior to inundation with tailings.

There is an opportunity to reduce tailings disposal costs by early mining of the closer satellite deposits (Turkey Creek and Bedford), and commence in-pit tailings disposal in these voids. This approach would require further environmental approvals, supported by detailed groundwater modelling.

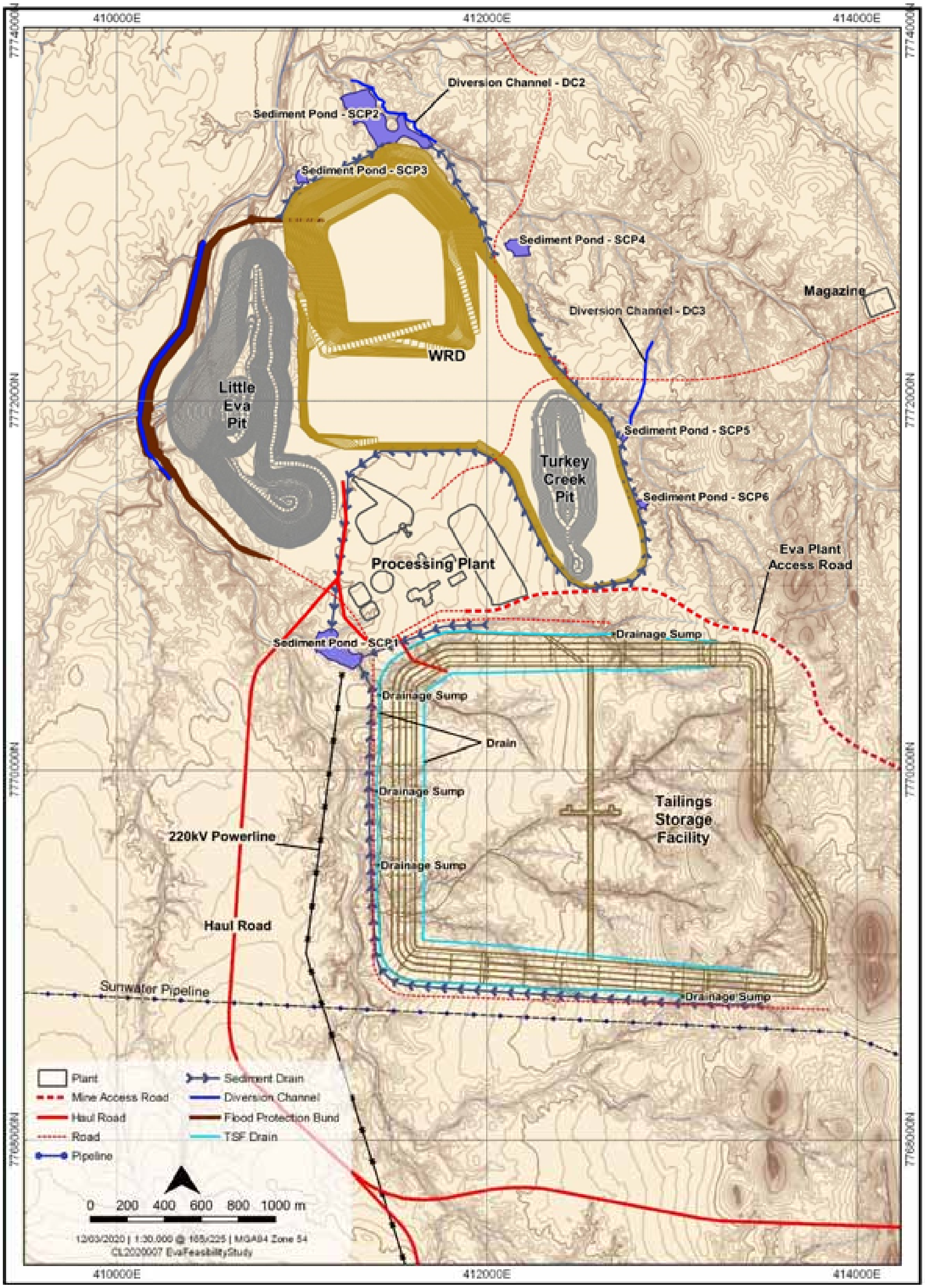


Figure 18-14: Tailings Storage Facility Site Arrangement

18.5.3 Testwork

18.5.3.1 Geotechnical Investigation

A geotechnical investigation of the proposed TSF and plant site was carried out by Knight Piésold in 2018, and consisted of the following:

- Drilling of four boreholes in the processing plant area, and three within the TSF embankment footprint
- Test pits at 35 locations throughout the western side of the TSF basin
- Laboratory testing of selected samples.

A summary of laboratory testing results for the TSF is as follows:

- The tested samples classified predominantly as sandy/gravelly clays to clayey/silty sandy gravels of low plasticity. Fines content (passing 75 μm) of between 16% and 58% were measured, indicating low to medium clay content.
- Falling head permeability tests conducted on remoulded test pit samples yielded permeabilities of between 8.6 x 10-9 and 3.7 x 10-8 m/sec compacted to 98% standard maximum dry density (SMDD) and at optimum moisture content (OMC).
- Dispersion testing on selected samples (KPTP-02, KPTP-09, KPTP-13, and KPTP-20) yielded results of Emerson Aggregate Test Class 4. The tests imply that the materials should be non-dispersive when exposed to the effects of stormwater events.

Based on interpretation of the investigation findings and testing data, the following conclusions can be made:

- It was considered that approximately 50% of the surface materials within the TSF should be suitable for rework as a low-permeability soil liner.
- Borrow material may be sourced from the alluvial and residual soil horizons that are present across the site for Zone A, Zone C, and general backfill materials.

18.5.3.2 Tailings Laboratory Testing

Testwork was carried out on tailings samples from the Project by Knight Piésold (2006), KCB (2019), and Paterson and Cooke (2019) to determine settling characteristics and expected decant returns, and included the following:

- Particle size distribution of the tailings
- Supernatant liquor density
- Liquid and plastic limits of the tailings solids
- Tailings solids particle density
- Undrained and drained sedimentation tests
- Air drying tests
- Permeability tests
- Rowe cell and consolidation tests
- Viscosity tests
- Tailings geochemistry.

Key testwork results included the following:

- The grading curve indicates that the sample is uniformly graded. Testing by KCB in 2019 found that the Turkey Creek, Blackard, and Little Eva tailings gradations were typical of Coarse/Hard Rock tailings using the International Commission on Large Dams (ICOLD) classification system.
- KCB testing found the Turkey Creek and Little Eva tailings are composed of an approximately equal mixture of sand and silt, consisting of 51% sand, 42% silt, and 7% clay sized material. The Blackard tailings stream is finer, with 13% sand, 76% silt and 11% clay.
- The tailings are generally classified as low-plastic sandy silt (ML-Unified Soil Class) under the Unified Soil Classification System (USCS).
- Knight Piésold testing in 2006 found that the tailings achieved a maximum dry density of 1.64 t/m^3. This is a moderate to good dry density for sandy silt tailings. Assuming the TSF is efficiently operated, it is estimated that the average field density of settled tailings will be between 1.55 and 1.63 t/m^3. Subsequent consolidation modelling from KCB in 2019 using Rowe cell test data gave a recommended design dry density of 1.6 t/m^3.
- Based on testwork on the Little Eva ore, the resulting tailings are classified as non-acid generating. The tailings generated will have significant acid neutralizing capacity.

18.5.3.3 Geochemistry Testing

Tailings fluids contain very low concentrations of metals, salts, and metalloids. Water leachate testing on blends of tailings from the Little Eva deposit with those from the Blackard and Scanlan deposits in 2011, and only the Little Eva deposit in 2012, were found to be slightly alkaline (pH values of 7.8 to 9.0), brackish, and contained very low concentrations of soluble metals. The composition of the major ions and metal ions within the water leachates were comparable to those of underlying groundwater in the mineralized areas. MBS Environmental in 2011 predicted that long-term weathering would have minimal impacts on the water quality of seepage from tailings, due to buffering properties of the carbonate minerals in the ore and waste rock. The results indicate that seepage from tailings, if it did occur, would not result in significant environmental harm.

18.5.4 Design Objectives and Parameters

The design objectives for the TSF are as follows:

- Permanent and secure containment of design tailings quantities
- Reuse of free water
- Reduction of seepage
- Containment of design storm events within the TSF
- Ease of operation
- Progressive and effective rehabilitation and closure.

A consequence category assessment for the Eva TSF was performed by KCB following both Queensland DES (2016) and ANCOLD (2019) guidelines. The results of the assessment are as follows:

- DES (2016)—In the event of a dam break, the Eva TSF has been assessed to be a High consequence structure according to the DES guidelines based on the General Environmental

Harm category. The structure is therefore a regulated structure. In the event of failure to contain overtopping, the consequence category is Significant.

- ANCOLD (2019)—The estimated Potential Loss of Life (PLL) in the event of a dam break is between 5 to 50. The PLL score, combined with the damage classification of Major for Business Importance and Infrastructure, results in a dam consequence category of High A based on the ANCOLD guidelines.

The key design parameters are shown in Table 18-3.

Table 18-3: Tailings Storage Facility Design Parameters

Parameter	Value
Storage Capacity	
Stage 1 Stage 10 (final)	16.3 Mt (20 months) 170 Mt (15 years)*
Operational Freeboard	0.5 m (from top of beach to crest)
Tailings Properties	
Design Tailings Stream Blend Slurry Solids Concentration In-situ dry density Beach slope	Blackard (25%), Little Eva: (70%), Turkey Creek (5%) 63% 1.6 t/m 1%
Embankment Geometry	
Perimeter embankment (intermediate) Perimeter embankment (final) Decant causeway Perimeter embankment (upstream) Perimeter embankment (downstream) Decant causeway	28.0 m 18.0 m 14.0 m 2 H:1 V 2.75 H:1 V (intermediate), 3.5 H:1 V (overall closure) 2 H:1 V
Water Management	
DSA volume Mandatory reporting level/extreme storm storage Spillway design flood	1 in 20 AEP, two-month wet season volume Maximum of 1-in-100 AEP 72-hour design event for each TSF cell 1-in-10 AEP wind-wave runup allowance in each cell. PMF
Earthquake Loading	
Operating basis earthquake Maximum design earthquake Post-closure	1:1,000 AEP, PGA =0.03 g 1,10,000 AEP, PGA = 0.13 g = MCE MCE
Factors of Safety	
Static (no potential loss of containment) Static (potential loss of containment) Static (long term residual strength) Post-seismic	FOS ≥ 1.3 FOS ≥ 1.5 FOS ≥ 1.1 FOS ≥ 1.2

Notes: * The TSF was designed to be constructed in stages, with a final capacity of 91.3 Mt, as this was the total tonnage at the time of design. It is envisaged that expanding the proposed TSF capacity to 117 Mt can be achieved without significant modifications to the TSF general arrangement and design philosophy presented in the DFS.
FOS = factor of safety; PGA = peak ground acceleration; MCE = maximum credible earthquake.

Prior to operations, an operating manual and a facility management document will each be generated, outlining the approach to tailings deposition, operating safety, seepage monitoring, and embankment stability; the manual will include an emergency action plan. The performance of the TSF will be continuously monitored and managed to ensure structural integrity and achievement of deposition and water return requirements. As the facility is categorized as having a High hazard rating according to DEHP guidelines, annual inspections will be carried out by a suitably qualified third-party engineer.

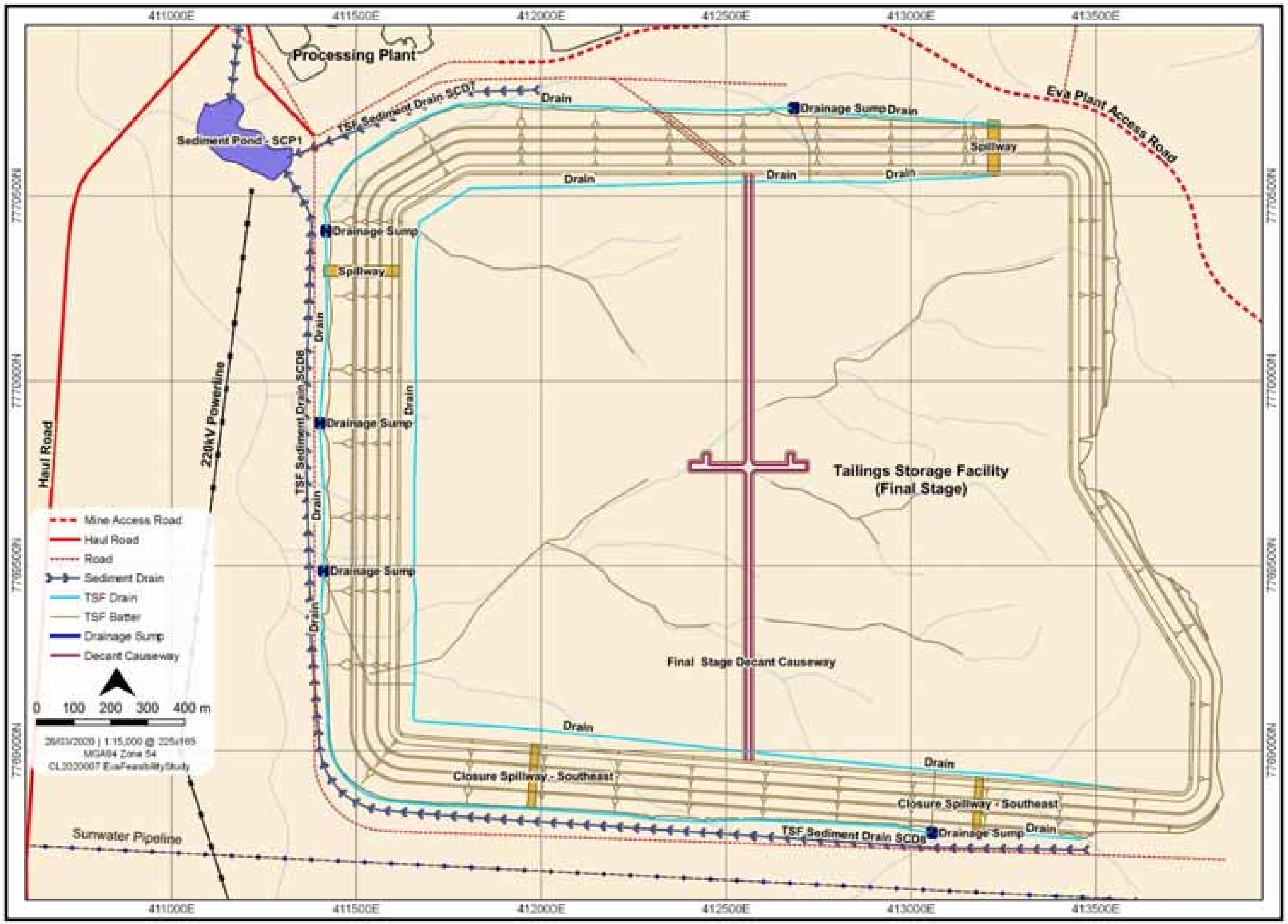


Figure 18-15: Tailings Storage Facility Layout Plan

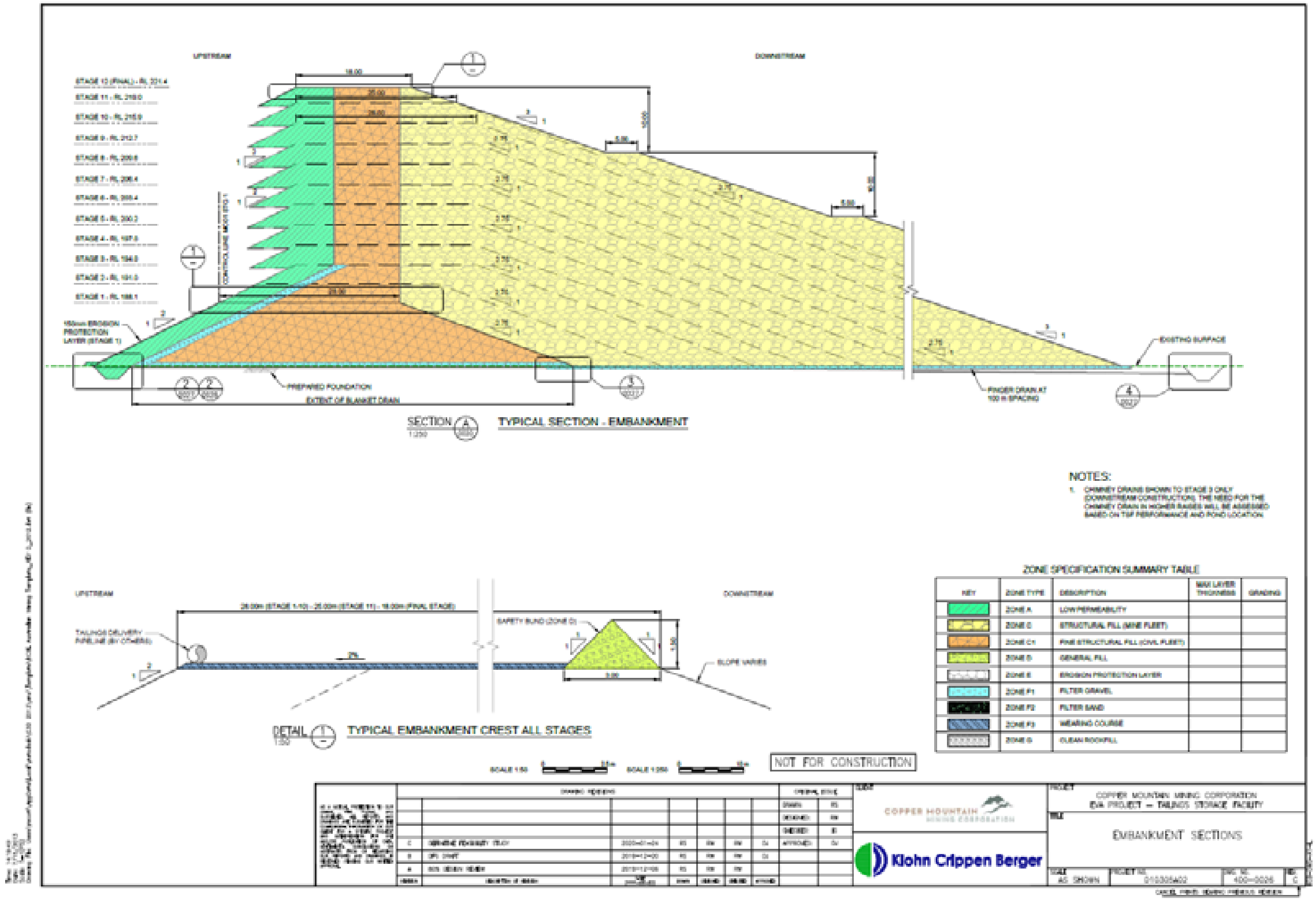


Figure 18-16: Tailings Storage Facility Typical Embankment Cross-Section

18.6 Accommodation Village

A permanent 300-person operations accommodation village will be established at a location 5.7 km east of the processing plant. The accommodation village complex will include:

- 75 four-room sleeping units (14.4 m by 3.3 m), including en suite air-conditioning rooms
- A single two-room disabled persons sleeping unit, including en suite air-conditioning rooms
- Dry and wet mess facilities
- Kitchen diner, cool room, and freezer room, 39 m long by 12 m wide
- Three laundries, including laundry machines
- Two toilet buildings (male, female, disabled), at the office and at the entertainment area
- One change room and toilets (male, female, disabled) at the recreation area
- An administration office, 18 m long by 12 m wide
- A bulk cleaning and cleaning materials storage
- First aid facility building, 12 m long by 3 m wide
- A recreation room ,18 m long by 12 m wide
- Swimming pool and sports court facility
- Communications/TV equipment facility
- Beer garden with shade sail awning
- Landscaping.

A sewage disposal system will be established at the accommodation village to treat up to 90 m^3/d of waste from the camp. An aerobic sewage treatment facility will be installed at the accommodation village to treat the waste water. Potable water for the accommodation village will be supplied from a water well hole to a tank and water treatment plant at a design rate of 90 m^3/d.

Early commitment to design and staged delivery of the permanent facilities will enable the construction workforce to be housed and messed in the permanent accommodation village.

There will also be a 150-person temporary construction camp installed early in the construction program, which will house the workforce prior to the permanent accommodation's village being operational and will remain onsite until the Project ramps down below the 300-person level. An initial workgroup carrying out establishment works will be housed regionally prior to commissioning the temporary construction camp. The workforce will then move to the accommodation village, with overflow residing in the temporary camp.

The Project is expected to directly employ some 460 people during the construction phase, reducing to around 280 during operations. Most of the construction workforce will be provided by contractors from the North West Queensland region.

During the operations phase, the workforce will be a combination of people living in Cloncurry commuting daily to site and FIFO people from the regional centres, which already provide personnel to the major mining centers in North West Queensland.

18.7 Site Buildings

18.7.1 Security Office and Tag In/Out Board Building

The employee work site tag in/out checkpoint will be located in a 12 m by 3 m building near the bus stop and parking lot at the plant site entrance. A canopy between these buildings will provide shelter for the site ambulance.

18.7.2 Administration Buildings

A centralized administration office will house the site management team, including general management, commercial and administration management, engineering, mine operations, senior processing, and maintenance personnel. The administration office will be a 30 m long by 25 m wide transportable building complex. It will comprise fifteen 14.4 m long by 3.45 m wide modules, one 3 m wide roof-over platform, and an additional 14.4 m long by 3 m wide area housing the ablution units and lunch room.

A first aid facility will be located at the administration office to service the plant site and mining activities. The paramedic will be stationed at this facility.

The administration office will also feature covered breezeways and verandas, open office partitions, split system air conditioning, floor coverings, electrical and communications services, hot/cold and waste water service, and a kitchen, complete with water boiler, refrigerator, sink cupboard, and tables and chairs. Specifically, the facility will accommodate:

- Standard enclosed offices and work stations for approximately 40 employees
- Partitioned work areas
- Three conference rooms
- Reception area
- Stationery and equipment storage
- Photocopy, document control, and mail room
- Information technology (IT) communications room/office
- Plotter and printer rooms
- Cleaners storage
- Lunch room equipped with microwave, fridge, sink, and dishwasher (5 m by 3 m)
- Three separate ablution units, one male, one female, one disabled, each unit 3 m by 8 m.

18.7.3 Control Rooms

A dust-proof, air-conditioned main control room and operations supervisor office will be provided above the grinding floor of the processing plant (adjacent to the crusher dump pocket) for operation of the primary crusher and rock breaker, providing a clear view of critical processing equipment.

There will also be a portable modular office located at the processing plant for the plant metallurgists and operations staff. Separate male and female toilet facilities will be located adjacent to the modular office.

18.7.4 Reagents Store Building

The reagents store structure will be a domed canvas structure on trusses, placed on sea containers, set on a concrete foundation. The PAX will be stored in a separate, built-for-purpose, ventilated and air-monitored steel container. These units will be fully serviceable by the site forklifts and loaders.

18.7.5 Assay Laboratory

A metallurgical laboratory will be established at the site. The laboratory will consist of a 19.6 m long by 12 m wide prefabricated wet laboratory building, comprising a wet chemical laboratory, chemical store, balance room, Atomic Adsorption Spectrometer (AAS), X-ray fluorescence (XRF) preparation and laboratory, instrument room, and four offices for the Senior Chemist and Metallurgist and Environmental Technicians. A 7 m by 10 m steel construction sample preparation area will be constructed adjacent to the wet laboratory area, with a covered walkway between the buildings. The buildings will be fully serviced with power, water, air conditioning and heating, air scrubbers, and fume hoods, and will be furnished.

Equipment to be installed in the laboratory includes:

- Pressure Filters
- Mill
- Workstation with splitter
- Crusher
- Pulverizers
- Sieve shaker
- Drying oven
- Dust collector
- Safety shower
- Freezer
- Fume cupboard
- Scrubber and scrubber fan
- Denver flotation machine
-
- AAS machine and exhaust fan
- Bond mill
- Silt trap
- Gilson screen
- Acetylene
- Waste collection tank
- Drain
- Mobile bench
- Nitrous oxide
- LECO Sulphur Determinator.

18.7.6 Concentrate Storage Shed and Weighbridge

A steel framed concentrate storage shed (approximately 35 m by 25 m) will contain the plate-and-frame pressure filter unit on the side, discharging dewatered concentrate onto the shed floor. Storage capacity within the shed will be nominally five days of concentrate production, with any excess to be stored in container laydown pads external to the building. The shed provides for CAT 988 loader access and a drive-through bay, designed for a road train with open-top concentrate-loading boxes, and allowing for two trailers to be loaded while positioned on the internal weighbridge (28 m long by 3.5 m wide). Once two trailers are loaded, the unit is driven forward to load the remaining trailers. Roll-up doors will be positioned at each end of the drive-through to assist with dust containment. A wheel wash and contaminant collection sump will be provided. There will be a small hut positioned in the weighbridge area to house electronics, including ticket printers and other scale hardware.

The concentrate storage shed must be demonstrated to meet Project EA condition such as dust control measures and others.

18.7.7 Mine Change House

The mine change house will be an 18 m long by 18 m wide transportable building complex. It will serve a total of 147 people and include dirty/clean locker facilities (75 men / 24 women), single clean-only lockers (39 men / 9 women), and shower and washroom facilities.

18.7.8 Truck Shop, Plant Workshop, Warehouse and Office

The truck shop and plant workshop/warehouse building will be a steel-framed building with overall dimensions of 68 m long by 30 m wide by 16 m high, with 12.4 m clearance to the crane rail. There will be six equipment bays located in the shop, each 9 m wide to provide ample clearance for servicing the mine equipment fleet. The shop will be equipped with a combined 50 tonne and 10 tonne bridge crane to provide for maintenance support. Compressed air will be supplied by a dedicated shop supply that will also supply air to the tire service bay located outside of the shop. Plant process equipment will have a provision for use of a service bay as an area for rebuilding larger production equipment under crane. Mechanical, electrical, instrumentation, and other maintenance and reliability engineering personnel will also have work benches provided for testing and servicing equipment.

The workshop area and the warehouse will be separated by a full-height internal dividing wall, and fitted with a 1.5 m high by 2 m wide roller door above, one personal access door, and a warehouse service counter. The warehouse will be fitted with heavy-duty pallet racking and high-density storage system units and shelving. There will also be secure locked storage to protect vendor-supplied consignment stock. The warehouse will have a receiving area and a covered storage rack for incoming direct charge purchases.

A 12 m by 3 m steel container will be positioned with hydraulic hose presses and fittings to provide for manufacture of hydraulic hoses to support production equipment. A second 12 m by 3 m steel container will store lubricants that will plumbed to the shop service bay.

Welding work will be done outside of the main shop in a separate domed structure that will be supported by a jib crane and a mobile crane.

An additional portable modular office complex (19.48 m long by 14.4 m wide) will be positioned to support mining operations, site-wide maintenance, and warehouse staff. Mine operations and the mine dispatch control center will also be in this building. A lunch crib and separate male and female ablution units with emergency wash showers will be located adjacent to this building.

18.7.9 Fuel Facilities

The Fuel Storage System will consist of two, 110 m^3 capacity, self-bunded tanks, complete with ladders and walkways, configured in a master/slave arrangement. Pipework, valves, filtration, and dispensing will be included. A dedicated unloading pump with high flow capacity will be provided to minimize unloading time. Light vehicle (LV) dispensing at 80 L/min will be provided. A heavy mining equipment (HME) dispensing package capable of 800 L/min will also be provided. SmartFill GEN 2 fuel management software and hardware will allocate and control fuel usage and ensure security and accuracy in fuel reconciliation. OPW automatic tank gauging will allow remote access to tank inventory levels.

18.7.10 Heavy Equipment Dispensing System

The unloading and dispensing areas will be situated at opposite ends of the fuel storage facility, so that filling of heavy and light vehicles can occur concurrently with diesel truck unloading.

Spill collection pavement, contaminant collection sump, and oil/water separation are included in the design. Electrical power will be supplied via an underground trench from the truck shop.

18.7.11 Fire Systems

An electric fire water pump, to be powered by the emergency standby system, will be installed adjacent to the fresh water/firewater tank. Suction to these pumps will be set up so that the lower compartment of the water tank provides dedicated fire water storage, as required by standards.

A ring main fire water system will be installed around the plant site, consisting of buried HDPE pipe, and steel risers and above-ground pipe. Fire hydrants and hose reels will be provided at strategic locations throughout the plant.

A fire control system and fire indicator panels will be established in specific locations throughout the plant.

Electrical switch rooms will be fitted with smoke detectors and alarms. Electrical power will be supplied via an underground trench.

18.7.12 Vehicle Wash-Down Facility

A vehicle wash-down facility will be provided, capable of cleaning heavy vehicles. The wash-down facility will be located to the northwest of the truck shop and the equipment ready line. System design will include fresh water supply piped to a high-pressure water pump, along with a buried oil/water separator. A purpose-built concrete slab will be designed to drain to the oil/water separator. Cleaned effluent from the separator will be returned to the process water streams in the process facility. Electrical power will be supplied via an underground trench from the truck shop.

18.7.13 Tire Services Pad and Service Area

A 20 m by 20 m concrete pad will be provided to service heavy equipment tires. In addition, there will be a 17 m wide by 12 m long by 10 m high dome and four steel containers, with one each functioning as an office set, tool crib room, parts storage, and liquids storage, to support a full-service tire maintenance contractor. Utility services of high-pressure air and electrical power will be supplied via an underground trench from the truck shop.

The tire service area will be established 275 m northwest of the truck shop workshop building. The tire services contractor will operate in compliance with safety exclusion zones, and have a safety gate and buzzer to restrict entry during tire inflating procedures.

18.7.14 Explosives Magazine

A mining explosives services provider will design, construct, license, and manage a compliant explosives magazine, and a bulk storage depot and handling facility for ammonium nitrate emulsion (ANE), ammonium nitrate (AN), and chemical additives (companion and gasser solution), in support of mobile manufacturing unit operations required for the Project.

The facility consists of the following features:

- AN system:
  - One 100 tonne capacity bulk-bag storage shed
  - One 1.2 tonne hopper
  - One forklift (with rhino horn attachment)
  - One electrically-powered auger
  - One 1.2 tonne tote bin (as backup, and for AN and ANE calibration).
- ANE system:
  - One 80 tonne capacity relocatable tank, with overhead load-out
  - One North Australian Pastoral Company (NAPCO) pump.
- Chemical additives system:
  - One 1,000 L companion solution storage tank (gravity-fed load-out)
  - One 1,000 L gasser solution storage tank (gravity-fed load-out).
- Process fuel oil system:
  - Process fuel will be supplied by the mine.
- Magazine Compound (minimum 175 m away from the explosives storage compound):
  - Magazines owned by Contractor, managed, and operated by the mine.
- Workshop/stores/spares facility:
  - One 20 ft long shipping container for archives
  - One 20 ft long shipping container for storage/spares.
- Water system:
  - One 10,000 L process water tank with camlock connection (outside fence)
  - One electric pump
  - Emergency shower and wash
  - Electrical system
  - One diesel generator.

The magazine will be established 3 km by road to the northeast of the plant site.

18.7.15 Mobile Equipment

Mobile equipment types and quantities are specified in Table 18-4. A front-end loader (FEL) to load concentrate trucks (Komatsu WA600 or equivalent size) will be required. Costs to transport the mobile equipment to the Project site are included in the infrastructure estimates.

Table 18-4: Mobile Equipment Details

Description	Quantity
Mine Fleet and Support	
Rotary LP Drill Atlas Copco PV 271 diesel 55 ft tower 1900SCFM Cat 345XL undercarriage 90 tonne op weight capable of 59 ft long single-pass drilling to 9.625 in to 10.625 in	2
Blasthole drill – Atlas Copco DM45, blasthole stemmer	1
Komatsu WA200PZ-6 or Cat 924 IT stemming wheel loader	1
Front loading shovel, Komatsu PC 4000, 22 m^3	3
Wheel loader, Komatsu WA900 13 m^3 bucket, high-lift	2
Haul truck, Komatsu HD-1500-7 dump truck, 144 tonne capacity(1)	22
Track dozer, Komatsu D375-A	4
Wheel dozer, Komatsu WA600-5	1
Komatsu 825A-2 motor grader	2
Ford truck 3133-H2O 4x6	1
Water truck, Komatsu HD605 63 tonne	2
Maintenance Equipment	
250 tonne mobile crane	1
80 tonne mobile crane	1
Shovel crew Hiab	2
Mechanics' tool truck	1
Fuel lube truck Komatsu HD605-7E0	1
Fuel/lube truck conversion (CAT 775B)	1
Front Cat 930G integrated tool carrier (Cat IT28 equivalent)	1
Wheel tractor scraper 24 yd^3	1
Telehandler zoom boom	1
40 ft boom elevating work platform	1
2 tonne capacity stores forklift	1
Komatsu WA600 tire handler	1
Light plants (10 x mining and 2 x plant)	12
Man lift (1 x mining and 2 x plant)	3
Elevated work platform (1 x mining and 1 x plant)	2
Line bucket truck	1
Portable steam cleaner (1 x mining and 1 x plant)	2
Forklift CAT IT28 site wide use	1
Shovel crew flat deck	1
Welder (Miller Big Blue)	2
Welder truck trailer	1
Electrical trade tooling and testing equipment	1

Description	Quantity
Maintenance plant shop tooling (1 x mining and 1 x plant)	2
Mine maintenance tool crib	1
Mine maintenance welding	1
Mill electrical trade tools	1
Portable wire compressor 185 CFM	2
Warehouse reach truck / Skid Steers x 2	1
Earthmoving and Support Equipment	
Excavator Komatsu PC850-8	1
Articulated dump truck Komatsu HM400-3MO; 1 x tailings and 4 x mine support	5
Wheel loader Komatsu WA600-5 (6.5 m^3 bucket); one for tip and one for concentrate loading	2
Screen plant and portable crushing unit	1
Tailings compactor Komatsu WF450-T (sheep foot tamping roller)	1
Skid steer loader (Bobcat)	2
Emergency Equipment	
Fire service and mine rescue trailer	1
Ambulance	1
Personnel Transport Vehicles	
Mine light vehicles for managers, safety, and maintenance	19
Mill light vehicles for mill manager, operations, and maintenance	9
Pit operations 12 passenger crew vans	2
Crew change 50 seat passenger bus	2

Note: Fleet brand names are used to provide indicative sizes, and do not necessarily represent equipment purchase commitments.
[(1)]Minimum 6, maximum 22, varies depending on year.

18.7.16 Transport

All of the equipment and materials required to construct the facility will be transported to site via road transport. International sea freight will either arrive at Townsville Port or the Port of Brisbane, from where it will be marshalled and dispatched to site as required. The company has engaged the services of an international freight forwarder to coordinate Project and operations freight requirements.

Site management and operating personnel will be transported to and from site from chartered roster flights via Company-owned buses between Cloncurry Airport and the accommodation village.

18.8 Communications

18.8.1 External Phone and Data Network Connection

The current communications plan was developed in 2012 for the original DFS. An update to the plan will be carried out during the detail design phase to ensure the Project benefits from advances in communications technology.

MMG have extended fibre optic cable (fibre) to the Dugald River Project from the Telstra facility in Cloncurry. The fibre from Dugald River to the Project will be installed and owned by the Project, and Telstra will provide services across this fibre to service the Project. This will entail an approximately 11 km run of fibre.

Telstra have established 4GX mobile phone coverage for MMG's Dugald River site, and this service will extend to provide mobile phone coverage for the Project.

A voice over Internet Protocol (VoIP) communications system will be installed.

18.8.2 Communications Rooms/Data Centre

Communications rooms will be established at the administration building and at the accommodation village. The communications room at the administration building will house the following equipment and systems:

- Fibre termination and associated network equipment
- Cabling termination for external fibre cable plant and UTP cabling within the administration building
- Data networking equipment
- IT servers and backup equipment
- Access control system server
- 15 kVA Uninterruptible power supply (UPS) and distribution.

The communications room will have the following infrastructure fitted by the building vendor:

- Smoke alarms connected to the building fire indicator panel
- Dedicated air conditioning (two split systems, 8 kW each)
- Five 240 VAC, 15 A power circuits
- One 415 VAC, 50 A circuit for a UPS feed
- Conduit access to external telecoms pit and pipe infrastructure.

The village communications room will house the following equipment and systems:

- Cabling termination for external fibre cables, internal Gigabit Passive Optical Network (GPON) fibre cables, and UTP cabling within the village administration areas
- Village corporate data networking equipment
- TV head end and distribution equipment
- UPS.

The village communications room will have the following infrastructure fitted by the building vendor:

- Smoke alarms connected to the building fire indicator panel
- Dedicated air conditioning (two split systems, 6 kW each)
- Two 240 VAC, 20 A power circuits
- One 240 VAC, 32 A circuit
- Conduit access to external telecoms pit and pipe infrastructure.

All power circuits will be fitted with appropriately dimensioned surge protection. The following equipment will be supplied and installed:

- One 3 kVA UPS and distribution
- Three 800 mm by 1,000 mm equipment racks.

18.8.3 Plant Tower Radio Base Site Shelter

A Radio Base Site (RBS) will be established at a location that will provide optimal radio coverage of the mine (pits), mine haul roads, waste dumps, ROM, TSF, administration, and plant areas. It is anticipated that this will be located in the area of the administration building. The final location will be selected in the detailed design phase.

18.8.4 Local Area Networks

A plant control local area network (LAN) will be established as part of the process plant control system. A switch and firewall interface to the network infrastructure will be provided at the administration communications room. The plant control system LAN will interconnect at this location via the fibre trunk.

The corporate LAN will be present in the administration building, plant store/workshop, and accommodation village. It will also be accessible from village rooms via a firewall, which will challenge users for access credentials. Only those users with the appropriate level of authority will be able to access the corporate LAN from the village rooms.

The accommodation village LAN will also be used by personnel for personal use. It will be accessible from village rooms via a GPON or wireless distribution system. The GPON system will also deliver TV to the village rooms.

18.8.5 Information Technology

IT infrastructure will include:

- Servers and backup devices
- Desktop computers, notebooks, network printers, etc.
- Desktop computing applications (including email client, Microsoft Office suite, BlueJeans cloud video conferencing, internet browser, Adobe Acrobat)
- Back office computing applications (including antivirus, anti-spam, databases, email server, and backup software)

- DISPATCH Fleet Management system for open pit mines for fleet optimization. Includes Positioning System (GPS)-based equipment, equipment health monitoring, maintenance tracking, blending, and production reporting.

Four categories of computers will be used across the Project: standard desktops, performance workstations, laptops, and servers. The operation and facilities will require a selection of standard (ST) and multifunction (MF) printers and plotters.

Table 18-5: Computing Equipment Requirements

Site Building/Facility	Desktops	Workstations	Laptops	Servers	ST Printers	MF Printers	Plotter
Administration Office	12	0	8	4	2	2	1
Plant Offices & Laboratory	20	-	7	2	1	1	-
Mine & Mine Workshop	14	4	6	3	2	1	1
Accommodation Village	2	-	1	-	1	1	-

Backup will be implemented using offsite cloud backup technology. Data will also be backed-up daily from each server hard disk to tape.

The Project software standard for desktop computing will be the Microsoft (MS) Office Suite.

Table 18-6: Software License Estimate

Desktop Software Product	Licenses
MS Office 365	76
MS Visio Pro	3
MS Project	2
Trend Micro NeatSuite client antivirus software	76
Adobe Acrobat Pro	18

Table 18-7: Server Software and License Estimate

Server Software Product	Licenses
MS Server	5
MS Enterprise Server Client Access	76
MS SharePoint Standard Client Access	0
MS SharePoint Enterprise Client Access	76
MS Remote Desktop Services Client Access	76
MS SQL Enterprise Server	2
MS SQL Server Client Access	76
MS Exchange Enterprise Server	1
MS Exchange Enterprise Client Access	76

18.8.6 Ultra-High Frequency Two-Way Radio System

The two-way radio system will comprise six Ultra-High Frequency (UHF) talk-through repeater channels with required coverage as per the channel coverage matrix (Table 18-8).

Table 18-8: Channel Coverage Matrix

Channel	Description	Plant/Admin.	Mine	Village	Tailings	Waste Dumps	Borefield
1	Emergency/Medical	✓	✓	✓	✓	✓	✓
2	Haulage/General	✓	✓	✓	✓	✓	✓
3	Mining	✓	✓	X	✓	✓	X
4	Drill and Blast	X	✓	X	X	✓	X
5	Maintenance	✓	✓	✓	✓	X	✓
6	Plant	✓	X	X	✓	X	✓
7	Simplex (to be allocated)						
8	Simplex (to be allocated)						
9	Simplex (to be allocated)						
10	Simplex (to be allocated)						

All channels will cover the site as far as is practical from the RBS. The mining and emergency/medic channels will be repeated into the respective pits as required when the receiving levels are affected by the pit depth. In addition, there will be several licensed simplex channels (nominally four). UHF CB channels can also be programmed to selected radios to allow communications with trucks transporting supplies to the mine.

Radio coverage analysis (based on un-mined terrain data) indicates that the site will be almost completely covered by the RBS installation. Coverage on the Burke Developmental Road is also expected to be very good. Once excavation at Little Eva exceeds 20 m in depth, a mine repeater trailer will be required near the pit edge to ensure UHF radio coverage throughout the pit and associated ramps. Radio transceiver equipment estimated to be required for the operation is summarized in Table 18-9 and has been included in the radio systems cost estimate.

Table 18-9: UHF Transceiver Quantity Estimate

Radio Type	Total
Handheld	100
LV mobile	22
HV mobile	30
Base station	17

18.8.7 Close-Circuit Television System

Closed-Circuit Television (CCTV) cameras will be installed at the security gatehouse to capture vehicle traffic passing through the gates. Other cameras have been allowed for fuel installations at the plant and mine areas if required.

18.8.8 Fire Control System

A fire control system based on NOTIFIER addressable technology will be implemented. Fire indicator panels will be installed at the following locations:

- Administration building
- Accommodation village
- Laboratory
- Crushing, grinding, and flotation switch rooms (three panels, one each)
- HV substation
- Mine workshop/admin
- Plant workshop/store
- Communications RBS shelter
- Fire control pump area.

The mill control room will house the master fire indicator panels. Village rooms have been excluded from the plant site central fire control system design, as these will be fitted with local smoke detectors and alarms that will report back to the mill control room through a single point of contact from the camp control panel, located in the camp office. Fire call points are included in the design.

18.8.9 Village Entertainment System

At the village, television content will be received via satellite. A DVD input channel has also been allowed for. This content will be modulated through the TV head end system, and distributed throughout the accommodation village (300 rooms plus common areas) via a GPON. This same GPON system will be used for the distribution of internet and telephone services. All rooms will be pre-cabled for TV via coaxial cable, and internet/telephone via Category 6 (CAT6) data cabling.

18.8.10 Telstra WAN (Data) and ISDN (Phone) Services

The wide-area network (WAN) for data will be delivered from the Telstra point of presence in the plant communications room. The WAN will consist of separate data services for corporate connectivity and village internet connectivity.

The Company’s corporate network will be a VoIP network cloud created by connecting the Project site and corporate office via Telstra VoIP WAN connections. Telstra will provide the WAN routers at each site, and the firewall services between the VoIP network and the internet, as managed services.

The telephone services for the Project will be provided by an Integrated Services Digital Network (ISDN) 10/20/30 Primary Rate Interface, which will also terminate at the plant communications room on the Private Branch Exchange (PBX).

All telephony services will be delivered via the LAN and the village GPON.

The communications infrastructure network will primarily be formed using CAT6 and single-mode optic cabling, interconnected via patch panels and communications closets strategically located within the complex.

High-speed data services will be delivered by fibre optic cabling (single mode) to primary ethernet switches within each communications closet.

A patch block system will be implemented in each closet so that office moves within each section of office spaces can be easily implemented. Voice/data cabling from each closet to office will be CAT6, certified to carry gigabit ethernet.

Voice and data outlets will be integrated into the same cabling standard from the office points to the communications wiring closets. Standard and managers' offices will have four shared voice/data points. Specialized rooms, such as video conferencing, printing, and training rooms, will have from four to eight points, depending on requirements.

18.9 Offsite Infrastructure (if required)

CMMPL has a five-year concentrate off-take agreement with Glencore International AG that may be extended for another five years by mutual agreement. The concentrate will be loaded by Glencore on site into sealed containers, and transported on the road by triple-road train trucks to Mount Isa mines at no cost to CMMPL. If the Glencore contract is not renewed, the following infrastructure is in place as a backup.

18.9.1 Cloncurry Storage and Rail Load-Out Facility

The QR National container yard and load-out facility at Cloncurry is equipped to store and load containers onto flatbed rail wagons for dispatch to Townsville port, and may serve as a Cloncurry container holding terminal for the Company if needed. Townsville Port Storage and Ship Load-Out Facility.

The Project has access to the Townsville Port QR National container storage yard. As further detailed in the Logistics Study report by Gilbride Management (2011), containers would be unloaded into ships' holds using QR National-owned rotating equipment. There is no requirement for the Project to provide initial capital to fund infrastructure at the port. Townsville Port will serve as a logistics hub and freight marshalling center during the Project construction stage.

19 MARKET STUDIES AND CONTRACTS

19.1 Concentrate Marketing

The Eva Copper Project will produce a copper concentrate with a LOM grade averaging 28% Cu and 2.61 g/dmt Au. The mine is expected to produce on average 163,000 dmt/a of copper concentrate over the LOM. The material will be considered a “clean concentrate” with no deleterious elements that would cause smelters to penalize the material.

An offtake agreement has been finalized, with Glencore International AG for a hundred percent (100%) of the mine’s output, with a fixed duration of five years and commencing with the start of mine production. The contract may be extended for a further five-year period, by mutual-agreement. The sale of the concentrate will be made on basis as freight carrier at (FCA) Seller’s mine gate.

Treatment and refining charges with fees paid to smelters by mines for converting the concentrate into refined copper, will be based on the annual prevailing market terms (annual benchmark) established between major international copper concentrate producers and major Japanese smelting companies. These charges will reflect current market fundamentals at the time of sale.

Discussions with other potential off-takers (smelting companies and concentrate trading companies) indicated interest in Eva concentrates should they become available at the end of the initial offtake agreement. The marketing cost assumptions are based on discussions with major smelters and concentrate trading companies and on the Company’s own views and experience in the copper concentrate market.

19.2 Copper Price Forecast

The lack of investment in copper mines and mine expansions lead many analysts to believe that there will be a tighter market for copper concentrates well into the 2020s. On the other hand, forecasted world copper demand, fuelled by electronic vehicles and renewable energy, is expected to see growth well into the future. The increase in demand and the lack of commitment on the supply side tends to give support to the copper price (Table 19-1).

Table 19-1: Copper Price Full Cost

	2020	2021	2022	2023	Long Term
Copper Prices ($/lb)	2.84	2.89	2.97	3.03	3.04

Source: CIBC Global Mining Group – Consensus Commodity Price Forecasts February 28, 2020.

19.3 Smelter Charges

Copper concentrates are sold by mines to smelting companies and merchants who charge treatment and refining charges (TC/RCs) to process the material. TC/RCs increase in an over supplied market and decrease when concentrate availability is tight. Treatment charges are calculated per dry tonne (dmt) of concentrate and refining charges are calculated per pound of payable copper. Consensus points to a tight concentrate market given the limited project development as well as expected smelter expansion required to meet the copper demand. This is especially true in China where deficits are forecasted for the next several years.

Table 19-2: Smelter Charges

	2020 Benchmark	2021 Forecast	2022 Forecast	2023 Forecast	Long Term Forecast
Treatment Charges ($/dmt)	62.00	75.00	75.00	75.00	76.00
Refining Charges (¢/lb)	6.20	7.50	7.50	7.50	7.60
Total TC/RC (¢/lb)	**16.62**	**20.10**	**20.10**	**20.10**	**20.37**

Other terms used in the study are internationally recognized standards for copper and precious metal payables and precious metal refining charges.

- Copper: 96.5% with a minimum 1-unit deduction
- Gold: 92.0% with gold content between 3 g/t and 5 g/t; and
  94.0% with gold content between 5 g/t and 7 g/t

The typical refining charge for gold at this grade range is $5/oz.

19.4 Precious Metal Forecast

Table 19-3 shows the precious metal prices obtain from CIBC Global Mining Group—Consensus Commodity Price Forecasts February 28, 2020.

Table 19-3: Precious Metal Price Forecast

	2020	2021	2022	2023	Long Term
Gold ($/oz)	1,521	1,507	1,466	1,434	1,362

19.5 Concentrate Markets

With a long-term off-take agreement now in place Eva copper concentrates are fully committed for the first five years of production; however, if the contract is not extended past the present agreement, other markets would be readily available.

The copper concentrate market is predicted to move to a deficit position in the next few years as global copper concentrate output is expected to grow at a slower rate, making it difficult to meet demand of expanded smelting capacities. China is expected to continue to expand its smelting capacity and although there are no firm smelter projects outside China, additional smelter capacity in countries such as Indonesia, Iran, Mongolia, and Zambia, are strong candidates for potential recipients of Eva Copper Project concentrates. Governments in developing economies that have mine production are also looking for additional concentrates to ensure enough smelting capacity to treat concentrates locally.

Should the initial sale and purchase agreement not be extended, the clean concentrates produced at the Eva mine would have no trouble finding a home in Asian smelters or with international trading companies.

19.6 Royalties

State of Queensland royalties apply to all lands except freehold claims prior to 1904. State Royalties range between 2.5% and 5.0% of metal value, less certain allowable expenses. If the concentrate is processed in Queensland (Mount Isa) there is a 20% reduction in the copper royalty. 100% of the royalty savings from the Queensland Government is for the account of the Seller (Copper Mountain Mining Corp. [CMMC]). Royalties are discussed in detail in Section 4.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Approval Process and Status

Mining projects in Queensland require:

- Tenure (Mining Leases [MLs]) from the Department of Natural Resources, Mines and Energy (DNRME) that gives access to the land.
- An Environmental Authority (EA), from the Department of Environment and Science (DES) that regulates the environmental management of the Project.

Both the MLs and the EA have been approved. The EA has been subject to a number of amendments, and the history is summarized in Table 20-1. The initial EA application process included a voluntary Environmental Impact Statement (EIS) and Environmental Management Plan (EMP). A Major Amendment application to the EA was submitted and approved in 2016, and reflects the 2017 DFS mine layout and plan.

Table 20-1: Project Environmental Authority (EA EPML00899613) History

Grant Date	EA Number	Project Basis	Comment
Jul 2012	MIN102973311	2009 DFS, which include pits at Little Eva, Blackard, and Scanlan.	Voluntary EIS submitted 2007, approved July 2008. EIS supplementary report 2007. EIS response to information request 2008. EMP prepared 2011.
Mar 2013	MIN102973311	As above.	Deferred Project start date and submission of management plans and programs to prior to commencement of mining activities.
Oct 2013	EPML00899613	As above.	Deferred all remaining significant due dates to prior to commencement of mining activities.
Jul 2016	EPML00899613	2017 DFS mine plan/layout removed Blackard and Scanlan, expanded Little Eva pit, added Lady Clayre, Bedford, and Turkey Creek, deposits, relocated the tailings storage facility (TSF) and added the Cabbage Tree Creek diversion channel and dike (bund).	Current Includes new environmental offsets framework introduced in Queensland on July 1, 2014.

Changes made to the mine layout in this Feasibility Study require a new amendment to the existing EA. Amendments are assessed to determine whether they are classified as Minor or Major. The extent of the new mine footprint, increased processing throughputs, adjustments to the waste dump, plant areas, TSF, Cabbage Tree Creek water well field, and road routes, and inclusion of the Blackard and Scanlan deposits to the mine plan will require submission of a Major Amendment Application to the existing EA. From the date of application submission, the Minor Amendment process takes up to 35 days, while the time for a Major Amendment can vary. The 2016 Major Amendment by Altona took 3.5 months from the date of application submission.

Sustainable Mining Strategies (SMS) and MBS are engaged as environmental consultants to Copper Mountain Mining Corp. (CMMC), advising, managing environmental surveys and EA submissions, and providing support with the collection and preparation of routine baseline monitoring prescribed by the EA.

The EA sets out key environmental management conditions and should be referred to for full details. MBS provided memos outlining key secondary approvals and environmental offset considerations, since which the Queensland Government introduced rehabilitation and FA reforms. These reforms introduced subsequent to approval of the current EA and previous Feasibility Study included the *Mineral and Energy Resources (Financial Provisioning) Act 2018* (*MERFP Act*), which was passed in November 2018. New regulatory requirements result from the reforms and have been considered.

Key EA regulatory management issues, particularly in the mine development period, are:

- EA Major Amendment Application—The current EA is based on a previous 2016 mine layout. Changes to the mine layout will require submission of an EA Major Amendment Application to the DES. This is a straightforward requirement that, with application preparation and pre-lodgement meetings.
- Progressive Rehabilitation and Closure (PRC) Plan Submission—Organizations carrying out mining activities in Queensland are legally obligated to rehabilitate the land. Recent legislation reforms require holders of an existing EA for a mining activity relating to a ML approved through a site-specific application granted prior to introduction of the PRC plan requirement (as per Eva), to develop and submit a PRC plan to the DES. As mine development at Eva has not commenced, a PRC plan is required to be submitted in conjunction with the proposed EA Major Amendment Application.
- Estimated Rehabilitation Cost (ERC) Decision—An ERC decision is required to be in effect before commencing any activities under the EA. The ERC is the estimated cost of rehabilitating the land on which a resource activity is carried out, and preventing or minimizing environmental harm, or rehabilitating or restoring the environment, in relation to the resource activity. DES is responsible for deciding the ERC for an EA for resource activities. The ERC came into effect in 2019 under the MERFP Act reforms, and replaces previous Plan of Operations (PoO) requirements. An ERC decision period may be of one to five years.
- ERC Scheme Financial Assurance (FA)—FA is required to be lodged with DES (either a contribution paid to the scheme fund, or a surety given under the MERFP Act) prior to any activities being allowed to commence. The amount of the FA required is calculated in accordance with DES procedures, based on the implementation of site-specific rehabilitation and closure tasks, using independent third-party contractor rates. The amount of the FA is directly related to the activities planned in ERC decision period.
- Design Plan for Cabbage Tree Creek Diversion—The necessary work has been undertaken by Knight Piésold; however, final detailed plans will need to be formally submitted and approval received prior to construction being allowed to commence.
- Environmental Offset Requirements—The Project triggers the requirement of an offset due to disturbance of regional ecosystems resulting from the disturbance of Cabbage Tree Creek. There are two options for offsets: a financial settlement, or a proponent-driven offset, which may include approved conservation work programs. A series of submissions are required, including Significant

Impact Details, Offset Report, and Notice of Election at least four months prior to commencement of any site work (Significant Residual Impacts). To fulfil its obligations, the Company intends to opt for a financial settlement, but is interested in investigating a proponent-driven offset (at least in part) involving the rehabilitation of Cabbage Tree Creek utilizing an indigenous contractor.

20.2 Statutory Requirements

State environmental legislation relevant to the Project includes:

- *Environmental Protection Act* 1994 and associated Environmental Protection Policies (EPP) including Environmental Protection (Air) Policy 2019, Environmental Protection (Noise) Policy 2019, and Environmental Protection (Water and Wetland Biodiversity) Policy 2019
- Environmental Protection Regulation 2019
- Environmental Protection (Waste Management) Regulation 2000
- *Environmental Offsets Act* 2014
- Environmental Offsets Regulation 2014
- *Mineral Resources Act* 1989
- *Mineral and Energy Resources (Financial Provisioning) Act* 2018
- *Water Act* 2000
- Water Regulation 2002
- Water Plan (Great Artesian Basin and Other Regional Aquifers) 2017
- Water Plan (Gulf) 2007
- *Nature Conservation Act* 1992
- Nature Conservation (Wildlife) Regulation 2006
- *Vegetation Management Act* 1999
- *Land Protection (Pest and Stock Route Management) Act* 2002
- *Aboriginal Cultural Heritage Act* 2003
- *Queensland Heritage Act* 1992
- Transport Infrastructure Act 1994.

Federal environmental legislation relevant to the Project includes:

- *Environment Protection and Biodiversity Conservation Act* 1999
- *Native Title Act* 1993
- *National Greenhouse and Energy Reporting Act* 2007
- *Energy Efficiency Opportunities Act* 2006.

20.3 Environmental Status

Supporting flora and fauna surveys, waste and tailings rock characterizations have been undertaken. This work was done to support the mining of all the deposits, location of the TSF, Cabbage Tree Creek diversion bund and channel, and new mining lease access road.

From the flora and fauna surveys, the key management issues relate to the following regional ecosystems:

- 1.3.7d (endangered)
- 1.3.6a (of concern)
- 2.9.4x41a (of concern).

These areas were mapped by the site 2016 and 2019 Vegetation Surveys (Figure 22-1), and it was determined that vegetation clearing in these areas triggers the above-mentioned offset requirement.

The area surrounding the Project processing facility site is uninhabited land, with the closest sensitive receptor being Mount Roseby homestead, approximately 17.5 km southeast of Little Eva, and 1.1 km from the nearest open pit at Scanlan. Noise and air quality monitoring is a requirement of the EA, and monitoring programs must be in place prior to the commencement of mining activities. Dust deposition monitoring was intermittently conducted at the Project site from January 2003 to October 2008. Results show that regulatory (EPP Air) standards have been exceeded locally on several occasions, unrelated to mining activities.

Tailings and waste characterization work summarized in previous chapters has shown both to be geochemically benign. Water and sediment management will require surface water and groundwater monitoring programs prior to commencement of mining activities. A Receiving Environment Monitoring Program (REMP) must be implemented at least six months prior to the commencement of mining activities.

Baseline water and sediment quality monitoring programs have been in place since 2012. Surface sample sites and water monitoring reference bores were prescribed in the original EA, which was based on the 2009 mine plan that included Blackard and Scanlan and a different TSF location. Monitoring locations were updated with the 2016 EA amendment. Monitoring sites made redundant by the removal of the Blackard and Scanlan areas from the EA amendment granted in 2016 were either retired in 2016 or reallocated to the newly added Bedford and Lady Clayre monitoring areas. Additional sites were also added to the new areas as required. Extensive baseline data from the Blackard and Scanlan sites was collected from 2012 to 2016, and monitoring from the sites will be re-established ahead of mining.

Additional reference bores required as a condition of the 2016 EA were established in June 2018 for the TSF and processing plant areas, and for the Little Eva deposit. Reference bores not required by the EA were also established at the Turkey Creek deposit. Reference bores at Bedford and Lady Clayre required as a condition of the EA have yet to be established. Six months of monitoring data from the reference bores set-out in the EA is required prior to mine development.

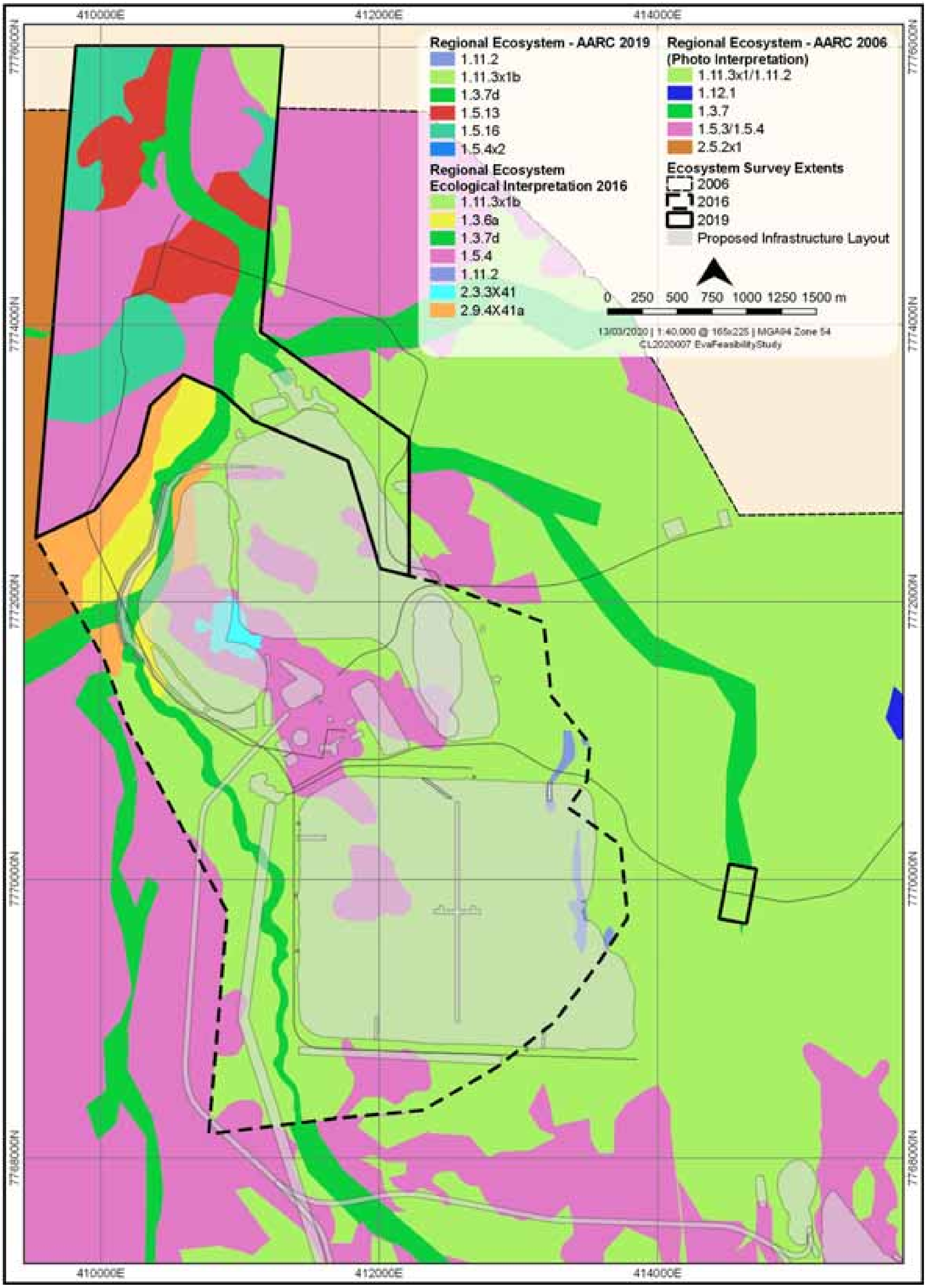


Figure 20-1: Regional Ecosystems Mapped in the Little Eva Pit Area and Site Layout

20.4 Environmental Risk Assessment

The key risks associated with release of contaminants into the environment have been considered. Surface water management designs incorporate environmental control drains and dams, developed to enable capture and recirculation of contaminated solids and liquids caused by potential spillage or due to site runoff. Even though tailings have been classified as geochemically benign, the TSF design includes a low permeability basin, cut-off drains, and monitoring bores to mitigate the risk of tailings liquor release into the local groundwater supply. The mine waste rock has also been classified as benign, and waste dumps will be rehabilitated to ensure revegetation of the area can be established, with contact water also managed with environmental runoff and sediment control drains and dams.

20.5 Cultural Heritage

The proposed Little Eva plant and pits, with minor area excisions, sit entirely within the Native Title area of the Kalkadoon People. Required for grant of the MLs, the Company has an Ancillary Agreement over the 143 km^2 area comprising the site MLs negotiated between the Company (then Universal and Bolnisi Logistics Pty. Ltd.) and the Kalkadoon Claimants on behalf of the Kalkadoon People, and signed on June 15, 2006. The Kalkadoon People and the Company (then Altona and Roseby Copper Pty. Ltd.) also signed an Ancillary Agreement (Agreement 909) with respect to all the Exploration Permit for Minerals (EPM) within the Kalkadoon Native Title Area on October 16, 2015. This area fully incorporates the Project EPMs, and provides access for exploration only.

Cultural heritage clearances over most of the area of the MLs, except for the western portions of ML 90164 and ML 90165, have been conducted. The ML Ancillary Agreement came into effect on June 15, 2006, and includes maps indicating the status of the cultural heritage surveys, and the sites of Cultural Heritage Finds (CHF), as at the start date of that agreement. CHFs from all clearance surveys since conducted by Universal, Altona, and Xstrata are recorded and tracked in the Company database and considered in mine and Project development planning. One recorded CHF site at the Little Eva deposit requires relocation ahead of mine development. In the process of agreeing to a process for relocation of the recorded site, new CHFs were identified in the area of the Little Eva pit and the adjacent impacted flood plain (probably exposed by recent flooding and vegetation removal by stock). A scope of work has been developed to finalize the relocation of the documented site and check for and relocate any other CHFs in the area impacted by Little Eva pit. This work is to commence immediately, once weather permits, in the first half of 2020 following the wet season.

Relevant government agencies, stakeholders, and the general community were consulted during preparation of the Project EIS, and had the opportunity to comment on the Terms of Reference and the EIS. Further opportunities for public input to the Project were available during the development of the original Project's EA and the revised EA. The EA Major Amendment Application required as a result of this study will require further stakeholder and public consultation.

The evidence of European history in the area is not of local or state significance under the Local Authority policies or the *Queensland Heritage Act* 1992 (Qld) or the *Environment Protection and Biodiversity Conservation Act* 1999 (Cth).

20.6 Employees

The Company's objective is to build the capability of our workforce through the realization of the full potential of our employees, thereby sustaining our long-term success as a company. We will offer employees career opportunities, training, and development.

We do not tolerate any form of workplace discrimination, harassment, or physical assault.

Copper Mountain Mining Corp. (CMMC) is committed to employing local residents from the communities in which it operates. CMMC operations will be dependent on fly-in/fly-out (FIFO) and drive-in/drive-out (DIDO) employees. Preference will be given to potential employees from the local Cloncurry community; however, technical, and professional skills are not always available locally. All employees in professional and technical roles will be offered the option of relocating to Cloncurry at the Company's expense.

Through CMMC's agreement with the Kalkadoon people, CMMC will strive to provide employment opportunities for local indigenous people.

CMMC will seek to employ a diverse workforce, to benefit from a varied range of skills, backgrounds, and perspectives. CMMC strives to employ people based on the skills and experience required for each position, without discrimination according to gender, race, age, sexual orientation, religion, or nationality.

20.7 Community Relations

CMMC acknowledges the importance of the communities in which CMMC operates. CMMC strives to be valued as corporate citizens in our communities. CMMC respects the rights, values, and cultural heritage of local people wherever it operates.

The Company believes that when we build and operate the Eva Copper Project, CMMC must contribute to the development of the Cloncurry community, and engage with our stakeholders in an open dialogue to maintain our social license to operate.

CMMC will make every effort to identify and address any concerns of local stakeholders by working with them, especially those most affected by our operations.

CMMC will seek consultation with indigenous communities regarding any impact CMMC may have on their territories, and will respect the agreements CMMC has made with Native Title Holders.

Landowners, leaseholders, and state and local governments are key stakeholders. The key local stakeholders associated with the Project are:

- Landowner – Harold McMillan (Mt. Roseby Homestead)
- Landowner – North Australian Pastoral Company Pty. Ltd. (Coolullah Homestead)
- Kalkadoon People
- Commonwealth and Queensland State departments
- Cloncurry Shire Council.

Over the last number of years, the Company has been in regular contact with the above stakeholders. During the construction period, the construction manager will be responsible for liaison with the stakeholders to ensure that any issues that arise are dealt with in an appropriate manner. During operations, this will be the responsibility of the General Manager, who will create a Community Development Plan to cover aspects such as how personnel will engage with the Community, conflict resolution, method of allocation of funds to sponsor community events, training and academic scholarships for local people, and employment. The Plan will also include aspects specific to management of indigenous cultural heritage issues.

20.8 Traffic Management

The key community risk that will have to be managed from commencement of operations through the LOM will be the additional vehicular traffic on the Burke Developmental Road and additional traffic through Cloncurry.

It is estimated that operations will increase the heavy vehicle traffic load on the Burke Developmental Road by approximately 24 one-way vehicle movements per day. Where vehicles are proceeding to Mt. Isa, they will bypass Cloncurry.

Local site access roads and tracks will be signposted, and traffic managed in such a way that occasional traffic (such as tourists or pastoral vehicles) will be excluded from the mine site and heavy vehicle areas, and directed towards the Administration Facility.

A Traffic Management Plan will be generated prior to operations, including assessment of the driving competence and fitness of all personnel and the roadworthiness of all vehicles admitted to site. These initiatives will reduce the risk to the community associated with vehicle accidents on public roads.

The Project's Health, Safety, and Environment (HSE) management procedures will be generated prior to commencement of operations, and will detail containment and clean-up procedures for any spillage considered to pose a potential health or environmental risk. All vehicles delivering reagents and chemicals to site will carry material safety data sheets. A nominated spill clean-up team will have access to a mobile spill clean-up trailer to enable them to respond quickly to any spill emergency.

The concentrate produced on site will be carried to Cloncurry in sealed containers, eliminating the loss of concentrate in transit and reducing the potential for spillage in the event of an accident.

Most personnel will be accommodated on site during operations, which will minimize the number of small vehicles commuting to the site on the Burke Developmental Road.

20.9 Health and Safety

CMMC believes that safety is everyone's responsibility. In everything CMMC does, wherever we are, we put safety first every time. We aspire to achieve zero harm to people, and believe all accidents are preventable. CMMC continuously engages with employees, contractors, and relevant stakeholders on safety matters. CMMC's goal is to end a day's work with all personnel and the environment in the same condition as when they arrived, if not better.

We strive to create and sustain injury-free, safe work environments for everybody in our workplaces. Zero harm is CMMC's goal. To achieve this, CMMC makes its management accountable for safety performance, and trains our employees to improve their safety knowledge and skills, making them aware that they have a responsibility to themselves, their family, and friends to work and behave safely.

We also ensure that every task undertaken in our workplace has a safe system of work identified, and that our people have tools and equipment that are fit for purpose and well maintained, to complete their tasks safely and productively. Furthermore, CMMC will implement regular health screening programs to monitor our employees' health, wellbeing, and fitness for work.

20.10 Construction Health and Safety

The engineering, procurement, and construction management (EPCM) engineer selected to build the Project will have the responsibility to establish and monitor compliance with the safety systems and procedures applicable to the construction site. The EPCM engineer will generate a Health and Safety Management Plan (HSMP) that establishes the structure to ensure:

- A safe and healthy working environment
- Safe systems of work
- The use of safe plant, equipment, and work practices on site
- Continuous promotion of the awareness of health and safety issues.

The HSMP will specify procedures to be followed in relation to all construction activities to ensure that there is proper planning, risk assessment, hazard identification, review and approval, safety management, and compliance inspection. Some of the key procedures that will form a part of the HSMP are:

- Materials management
- Confined space entry
- Crane management
- Excavation procedure
- Hot work procedures
- Job hazard analysis
- Fall prevention and protection"
- Site entry and security
- Waste management procedures
- Welding health and safety
- Working at heights
- Electrical safety and lockout procedures.

During the construction period, the EPC engineers site management will be responsible for the induction of all personnel visiting the site.

20.11 Operational Health and Safety

The Owner's HSE Manager will develop a Health, Safety, and Environmental Plan (HSEP) for application to the Eva Copper Project operations. The plan will include operational procedures to safely manage all mining, processing, and maintenance activities on site, and through the logistical chain to Townsville. For specific activities (including mining, transport, and logistics) where contractors are providing services to the operation, the Owner will, after review and approval, incorporate their HSE procedures into the Owner's HSEP.

Once operations commence, the Owner will provide safety inductions to all personnel visiting the Project.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost

The initial Capital Cost Estimate (CAPEX) for the Project's process plant, power line, infrastructure, and mine was prepared in accordance with standard industry best practice for this level of study and to a level of definition and intended accuracy of ±15%. In some cases, the Feasibility Study improved on this level of accuracy because of the level of detail examined. The plant is a typical processing plant with few, if any, unusual features that are unknown or difficult to define.

The principal consultants, namely, Merit, Ausenco, KCB, Knight Piésold, together with Owner's consultant specialists in mine planning, geoscience, metallurgy, and environmental management provided input for the CAPEX. Each consultant provided a detailed set of quantities, assisted with the costs, were involved in the scheduling and strategy for execution, and met regularly to discuss the evolution of the study. Full participation and open communication amongst all parties occurred during the Feasibility Study process.

With the help of the engineering and consulting groups, Merit talked to contractors and service suppliers to establish today's market cost to as great a degree as possible. The exercise reviewed items in detail such as:

- Availability and cost of construction and operating personnel
- Productivities to be expected using information from a recently completed project in the same general area
- Locally available materials and support facilities
- Construction materials availability at site
- Work schedules and accommodations
- Use of some of the mining fleet for construction activities
- Areas of risk.

The initial project capital cost estimate provides for 1,600,000 total labour force hours, comprising 1,040,000 direct and 560,000 indirect hours, including pre-production personnel. The labour force at the Project site will peak at approximately 450.

The initial capital cost estimate covers the direct and indirect costs encompassing all of the traditional items that are standard to any project, and incorporates thinking based on similar projects that have been, or are being built over the last several years in and around the Province of British Columbia in Canada, and with some reference to other off-shore projects that have similar process plant sizes. In addition, the Feasibility Study consultants are currently involved with a number of similar size and type of other feasibility projects and therefore, much information was made available by way of group experience and knowledge. For example, Merit provided Construction Management services for the Copper Mountain Project located near Princeton, BC. Also, the principal consultants Ausenco, KCB, and KP, and several contractors that provided pricing information, have knowledge and experience from working in Australia. This first-hand knowledge was invaluable when verifying construction costs and methods.

This Feasibility Study initial capital cost estimate is considered to be an AACE Class 3 Feasibility Study estimate and has been compiled by Merit. It also presents the approach, methodology, format, and scope of the basis that were used for estimating the CAPEX, and provides guidance notes and key assumptions used by the engineering team.

21.1.1 Project Summary and Contributors

The initial Project capital cost estimate comprises four main categories:

- Direct costs
- Indirect costs
- Contingency
- Owner's costs.

The basis for the initial capital was developed by the following companies:

- Copper Mountain Mining Corp. (CMMC) (mine development, mine equipment, and mine infrastructure)
- Ausenco (process plant design)
- Ausenco (process equipment selection)
- CMMC (process equipment pricing)
- CMMC (ancillaries and plant site infrastructure facilities selection and pricing)
- Ausenco (plant infrastructure civil design)
- KCB (tailings storage and water management)
- Knight Piésold (pit diversion bund)
- CMMC (Owner's costs)
- Merit) pricing for process plant, and infrastructure construction; based on quantities, capital equipment, and building costs provided by the engineers and Owner)
- All Participants (Indirect cost contributions)
- All Participants (Contingency recommendation).

21.1.2 CAPEX Summary

The CAPEX was prepared in Australian dollars which were converted to United States dollars (US$) using a rate of 1.55 at the time of its preparation in Q1 2020, as summarized in Table 21-1.

The total initial capital cost US$382 million, as at Q1 2020 evaluation, subject to qualifications, assumptions, and exclusions, detailed herein. The total LOM development cost estimate includes all pre-production revenue and rehabilitation costs in Years –2 to 1, as shown in Table 21-1.

Table 21-1: Eva Copper Project LOM Capital Cost Summary

Capital Cost Items	Initial Years (Year -2 to Year 1) (US$ millions)	Year 2 to Year 15 (US$ millions)	Total CAPEX (US$ millions)
Direct Costs			
Mining	35.2	61.4	96.6
Process Plant	150.8	-	150.8
Infrastructure	67.6	-	67.6
Ancillaries	25.6	-	25.6
Total Direct Costs	**279.3**	**-**	**340.6**
Indirect Costs			
EPCM	25.1	-	25.1
Freight and Logistics	7.6	-	7.6
Indirect Costs	24.3	-	24.3
Owner's Costs	15.3	-	15.3
Total Indirect Costs	**72.3**	**-**	**72.3**
Subtotal	351.5	61.4	412.9
Contingency	41.5	-	41.5
Total Project Capital	**393.1**	**-**	**454.5(1)**
Pre-production revenues	(11.2)	-	(11.2)
Total Capital	**382.0**	**61.4**	**443.4**
Sustaining capital	-	34.0	34.0
Rehabilitation	1.28	12.9	14.1
Overall Project Capital	**383.3**	**108.2**	**491.5**

Note: (1)Total Project CAPEX is 704.5 in Australian dollars.

21.1.3 Estimate Structure

The Estimate structure has been assembled and coded with a hierarchical Work Breakdown Structure (WBS) by major area, area, sub-area, discipline, and optional coding system. The WBS structure is shown in Table 21-2.

Table 21-2: Capital Cost Estimate – Work Breakdown Structure

Level 1	Level 2	Level 3	Level 4	Level 5
300	320	321	231	00
Major Area	Area	Sub-Area	Discipline	Optional
Processing Plant	Grinding	Ball Mill & Cyclone	Concrete	Place Holder

All engineering groups have provided MTOs and pricing information in accordance with the Level 3 (Sub-Area) and Level 4 (Discipline) designations, as listed in the WBS. Where this was not practical and at the discretion and approval of CMMC, this requirement was relaxed to a Level 2 (Area) effort.

21.1.4 CAPEX Template

The CAPEX has been formatted and assembled in an Excel file. It is based on the finalized WBS, and takes into consideration the following:

The CAPEX document tabulated and summarized the information required for the final estimated costs, which include, but were not be limited to, quantities, productivities, labour rates, and materials, equipment, subcontractors' costs, indirect costs, Owner's costs, and contingencies.

To manage the inflow of information, the CAPEX template was the basis for all information required and submitted by all participants who contributed to the final estimate. The CAPEX document itself was under the control of Merit.

21.1.5 Responsibility Matrix

The Responsibility Matrix, based on the WBS, identified the required participant input of information for the full scope undertaken by CMMC (Owner), Merit (CAPEX), engineers (Ausenco, KCB, and Knight Piésold). These are summarized in Section 21.1.1.

21.2 Mining Capital Cost Estimation

Mining costs have been generated utilising the Owner's in-house operating experience, 4Q 2019 data updated with Queensland regional supplier input costs, and price quotations to develop capital and operating costs. The scope of the Owner's mining supply included all management, facilities, equipment, infrastructure, operators, maintenance, and administration required to mine multiple pits in accordance with the mining schedule to produce ore for processing.

The mining is described in Section 16 of this report and contains the detailed descriptions of the development methodology and equipment. A summary of the estimated costs for mine development, equipment and pre-production activities in Year -2 to Year-1 is summarized in Table 21-3 and includes mine development of approximately 14.7 Mt of stripping, 13.5 Mt of waste, and 1.2 Mt of ore in the final pre-production year. Mining capital cost estimates include offsetting pre-production activities.

Table 21-3: Mining Capital Summary

Development Mining Categories	AU$ (millions)	US$ (millions)
Mine development	5.7	3.7
Mine equipment	134.6	86.9
Pre-stripping	6.6	4.2
Explosives storage	1.3	0.8
Open pit dewatering	1.6	1.0
Mining Capital Total	**149.8**	**96.6**

In addition, under Area 380 Site Earthworks, the capital cost estimate includes the following major earthwork projects, which will be built by the Eva Copper Project mine staff.

- Run-of-mine ore pad
- Bulk depot and powder magazine
- Process plant areas
- Raw water pond
- Process water pond
- Administration buildings
- Warehouse
- Security office
- Plant site roads
- Main access road
- CTC borefield road
- First aid site
- Training/lunch area
- Main substation
- Magazine access road
- Tailings storage facility (TSF) access road
- Accommodation village
- TSF site preparation
- CTC bund preparation
- TSF bulk haulage
- Little Eva Pit pre-stripping

21.3 Process Plant and Infrastructure Capital Cost Estimation

The CAPEX breakdown for the Process Plant and for Infrastructure and Ancillaries are provided in Table 21-4 and Table 21-5 in both AU$ and US$.

Table 21-4: Processing Plant Capital Summary

Categories	AU$ (millions)	US$ (millions)
Crushing, ore storage, and conveying	96.0	61.9
Grinding	31.1	20.1
Flotation and regrind	29.9	19.3
Concentrate handling	15.2	9.8
Reagents	4.0	2.6
Assay laboratory	2.0	1.3
Site earthworks	15.8	10.2
Tailings thickening	9.7	6.2
Tailings storage facility	28.9	18.6
Tailings reclaim water	0.9	0.6
Tailings pipelines	0.4	0.2
Total	**233.8**	**150.8**

Table 21-5: Infrastructure and Ancillaries Capital Summary

Categories	AU$ (millions)	US$ (millions)
Power and electrical distribution	61.7	39.8
Water systems	19.8	12.8
Sewerage	0.3	0.2
Communications and documents	5.4	3.5
Air systems	4.1	2.7
Plant mobile equipment	6.1	4.0
Access roads	7.4	4.8
Administration building and offices	2.6	1.7
Truck shop and truck wash	4.9	3.1
Accommodation village (operation camp)	31.0	20.0
Fuel storage and distribution	1.2	0.8
Total	**144.4**	**93.2**

21.4 Basis of Estimate

21.4.1 Cost Estimation

In deriving construction costs, Merit worked with general regional contractors to establish current market costs to as large a degree as possible. Merit stated the execution strategy to develop the construction costs. As the estimate information was compiled, Merit estimated the direct and indirect man-hours and the labour force loading distribution curve was then calculated.

The design throughput capacity of the plant is 31,200 t/d. The capital and installation cost estimate reflect recent competitive bids for new equipment from well known equipment suppliers to process this tonnage.

21.4.2 Measurement System

The Project cost estimates and supporting material take offs (MTOs) are reported in metric units.

21.4.3 Design Growth and Waste Allowances

All the quantities identified by the engineering MTOs are final. Any costs associated with design growth that has not been fully quantified through the MTO process are included in the contingency.

Unit rates include all costs inclusive of wastage.

21.4.4 Productivities

Productivities for installing equipment and materials were provided by experienced local and regional construction contractors familiar with the Project's location and local conditions.

21.4.5 Estimating Based on Percentages – Allowances

Industry standards were applied in the form of percentages, where Engineering where Engineering MTOs or definitive estimates were not available

21.4.6 Direct Costs

Direct Costs were based on the following information:

- Process flow diagrams, site layout and general arrangement drawings, equipment list, electrical single line diagrams, and drawings from similar projects
- Budget submissions for the design/supply of new major and secondary equipment provided by vendors in accordance with specifications and/or datasheets developed by the engineering consultants, as provided by CMMC.
- Prices for permanent materials based on contractor's quotations, in-house data, and current market conditions
- Geotechnical information and recommendations provided by Knight Piésold and KCB
- Material quantity take-offs provided by Ausenco, KCB, and Knight Piésold
- Labour rates provided by local and regional construction contractors, and are included in the unit rates
- Local and regional construction contractors familiar with the Project location and local conditions provided productivities and unit rates. Productivities that can be expected for this location have been based on measured results from other projects as well as in-house data
- Supply and installation prices from experienced vendors of pre-engineered and modular buildings as provided by CMMC.

21.4.7 Indirect Costs

Indirect costs include the following:

- Engineering, procurement, and construction management services (including travel expenses)
- Temporary construction facilities including worker's camp, secure lay-down areas, warehouses, etc.
- Temporary construction services including contractor's mobilization/demobilization cost (included in contractors' unit rates)
- Construction accommodation and catering
- Construction equipment (mostly included in contractors' unit rates)
- Freight
- Start-up and commissioning allowance
- Vendor representatives
- First fills and capital spares
- Third-party engineering
- Quality assurance
- Surveying
- Owner's costs.

21.4.8 Pricing

Several key pieces of equipment such as the grinding mill, primary crusher, high pressure grinding roll (HPGR), drive motors, and the accommodation village are based on budgetary quotations. Pricing for commodities or other processing equipment was based on budget quotations obtained from vendors and contractors for major equipment and unit rates plus or minus 15%. Budgetary quotations generally mean that indicative pricing was provided for specified equipment, materials, and productivity; however, no commitment was made to secure the equipment or materials at this price for a future date.

21.4.9 Taxes

Goods and services tax (GST) are excluded from the capital cost estimate.

21.4.10 Currency, Estimate Base Date, and Foreign Exchange

All project capital costs are in United States Dollars (US$) with the following provisions:

- Costs are based generally on Q4 2019 market conditions with no provision for escalation beyond this date.
- Costs submitted in other currencies have been converted to Australian dollars and then United States dollars at an exchange rate of 1.55:1 for the purposes of initial CAPEX preparation.
- No provision was made for variations in the currency exchange rates from those indicated. No provision was made for any taxes or fees applicable to currency exchanges.

21.4.11 Accuracy

The capital cost estimate, for the mine, process plant, tailing storage facility and infrastructure was prepared to a level of ±15%.

21.4.12 Project Execution Plan

The CAPEX is based on the assumption that CMMC will follow the Project execution plan described in Section 24. Any deviation from this plan may have an impact on both project schedule and costs.

21.4.13 Contingency

The contingency percentages are allowances for undefined items of work, which cannot be foreseen or described at the time the estimate was being completed due to the lack of complete, accurate, and detailed information.

21.4.14 Assumptions and Exclusions

The following assumptions are the basis for the capital cost estimates:

- The design is based on the flowsheet, mass balances, design criteria, and equipment list developed for the Feasibility Study update
- Suitably qualified and experienced construction labour will be available at the time of execution of the Project

- Construction work is based on unit and fixed price contracts (no cost plus or time and materials arrangements)
- Budget quotes from vendors for equipment and materials are valid to within ±5% of the purchase price
- Concrete aggregate and suitable backfill material will be available locally; the Owner's geotechnical consultants identified suitable areas
- Soil conditions will be adequate for foundation bearing pressures
- Construction activities will be carried out in a continuous program
- Labour productivities are valid and established with input from experienced contractors and Merits' in-house database of current projects
- Bulk materials such as cement, rebar, structural steel and plate, cable, cable tray, and piping are all readily available in the scheduled timeframe
- Capital equipment is available in the timeframes shown and delivery has been verified by the requisite supplier.

The following items are specifically excluded from the processing plant and infrastructure initial capital estimate:

- Interest during construction
- Cost of financing
- Escalation during construction
- Sustaining or deferred capital costs
- Costs associated with force majeure events
- Sunk costs
- Future scope changes
- Environmental studies, permitting and mitigation
- Costs for community relations and services
- Relocation or preservation costs, delays and redesign work associated with any antiquities or sacred sites
- Environmental monitoring
- Warehouse inventories other than initial fills and capital and start up spares
- Schedule delays and associated costs such as those caused by:
  - Unidentified ground conditions
  - Extraordinary climatic events, force majeure
  - Labour disputes
- Insurance, bonding, permits, and legal costs
- Schedule recovery or acceleration
- Customs duties
- Research and exploration drilling
- Closure costs
- Salvage values.

21.4.15 Project Direct Costs

21.4.15.1 Quantities and Unit Pricing

Engineering material take-offs were based on quantities derived by the engineering groups from project drawings, sketches, and similar projects.

21.4.15.2 Bulk Earthworks – Site and Tailing Storage Facility

Quantities were developed by the engineering groups based on preliminary design drawings, 3D model, and sketches as follows:

- Site Development and Roads – by Ausenco
- Site water management and sedimentation ponds – by Ausenco
- TSF – by KCB
- Eva Pit Diversion Bund – by Knight Piésold
- Raw water supply lines from the Cabbage Tree Creek wells and Little Eva pit dewatering – allowance by CMMC
- Civil design is based on Knight Piésold Report PE801-00052/14, Rev B October 2, 2018, Definitive Feasibility Study – Geotechnical Interpretative Report
- LiDAR data, December 2019, provided by CMMC
- Quantities are neat without design growth
- MTO excluded any allowance for wastage
- Takeoff Allowance for minor errors in measurements, takeoffs and rounding was included by Ausenco
- KCB, Knight Piésold, and Ausenco provided earthworks designs, MTOs, preliminary specifications, and construction quality assumptions.

The earthmoving unit rates for the plant and infrastructure were calculated based on pricing obtained from regional civil contractors who have knowledge of the conditions in the area, and, where applicable, by CMMC for earthworks tasks to be self-performed. The rates included the rental of earthmoving equipment, operators, fuel, and mobilization/demobilization costs. CMMC personnel estimated costs for tasks performed by the Owner's mining fleet.

It has been assumed, from geotechnical information for the site, that concrete aggregates, structural backfill, granular base, road base and sub-base will be supplied from the borrow pits established at the site. The unit costs associated with these materials include borrow pit development (crushing and screening) and transport costs.

Bulk earthworks at the TSF, plant and diversion bund utilising mine-supplied materials was priced based on CMMC self-performing delivery to stockpiles and where appropriate placement and compaction by mine equipment.

21.4.15.3 Concrete, Formwork, and Reinforcing Steel

Concrete quantities were determined by Ausenco taken off design sketches and preliminary drawings, 3D models, preliminary equipment lists, etc.

A takeoff allowance has been included for minor errors in measurements, take-offs and rounding.

Regional industrial contractors provided the unit rates for concrete placement and finishing. Most construction aggregates for structural fill and concrete will be sourced locally with an on-site batch plant located close to the new plant site.

Formwork was estimated for each type of concrete classification, and includes local supply, form oil, accessories, shoring, and stripping.

Reinforcing steel quantities were developed based on estimated weight per m^3 of concrete for each type of classification based on ratios provided by Ausenco, and includes the local supply of material, cutting, accessories and installation.

Regional contractors also provided the unit rates for formwork and reinforcing steel.

21.4.15.4 Structural Steel

Steel quantities were determined by Ausenco and the calculation is based on the mechanical layout requirement with reference to 3D layout models.

Allowance for steel connections, stiffener plates, and miscellaneous tertiary supports are included in the MTO allowance.

Conveyor support beams, conveyor trusses, trestles, conveyor platforms, and take-up station structure are parts of the conveyor vendor package. These items are excluded from the MTO quantities.

The steel unit rates include:

- Material supply, detailing, fabrication, and surface treatment where required
- Erection at site based on estimated installation man-hours and unit labour costs and includes final touch-up of surface coating
- Connection steel, welds, and bolts.

The steel supply and erection rates were based on regional contractor’s pricing.

21.4.15.5 Architectural

Ausenco provided a list of structures and pre-engineered buildings and their respective sizes.

CMMC provided pricing for pre-engineered or pre-fabricated modular buildings.

21.4.15.6 Construction Camp – Accommodation Village

There will be a 150-person temporary construction camp installed to house workers early in the construction program, and this will also be used to house overflow worker force during the peak manpower requirements in the second year of construction. Prior to the on-site camp installation, construction and support personnel will be housed in Cloncurry or at MMGs existing camp at the Dugald River mine site. Costs for the temporary contractor supplied construction camp and the 300-person permanent accommodation village are included in the CAPEX.

An area will be prepared at the plant site for a 300-person Accommodation Village to be constructed prior to the second year of construction. The Accommodation Village will accommodate construction workers, construction management staff and associated visitors such as vendor representatives and other project team members visiting on a casual basis. Owner will provide bussing services to-from the campsite to all construction workers.

It is expected that the power supply subcontractor will provide their own accommodations but in case no room and board are available, their construction workers were included in the manpower load for camp capacity.

All mine operating personnel for pre-production and operations will be housed in the local community or at the accommodation village.

21.4.15.7 Mechanical Equipment

CMMC itemized and priced all capital equipment in accordance with the flowsheets developed by Ausenco. A detailed equipment list in Excel format was developed by the engineer, based on the template provided by Merit, for ease of transfer to the CAPEX. The equipment list was completed with pricing, sorted by area, and included the equipment description, equipment size, supplier name, currency, spare parts, etc.

Installation unit rates were based on estimated costs supplied by regional general contractors.

Vendor representatives will be engaged to oversee the installation of the larger equipment.

Vendor costs including allowances for travel time, food and lodging for onsite support, commissioning and performance testing were included in indirect costs, under start-up, and commissioning.

21.4.15.8 Plate Work and Tanks

Weights, for plate work, metal liners, tanks, chutes, launders, and pump boxes were calculated in kilograms of steel by Ausenco, based on the Project 3D Model, and detailed fabrication drawings from past projects.

Rubber lining for pump boxes and chutes were provided on a square meter basis.

Abrasion resistant (AR) wear plate and rubber linings were included as appropriate, and MTOs were calculated from design sketches based on preliminary layouts.

The unit rates include locally available plate purchase, detailing, fabrication, and installation based on regional contractor’s pricing.

21.4.15.9 Building Services

Heating, ventilation, and air conditioning (HVAC) equipment and ductwork, dust collection systems, fire suppression systems and miscellaneous building services piping allowances were provided by Merit from in-house historical data where required, and by CMMC where building pricing was provided.

21.4.15.10 Plant Mobile Equipment

The plant mobile equipment list with pricing was provided by CMMC.

21.4.15.11 Piping

Process piping allowances were based on a percentage of the mechanical equipment installed costs. Based on Ausenco in-house historical data, the piping distribution percentages included for all process areas is shown in Table 21-6.

Table 21-6: Capital Cost Estimate – Piping Allowances

Area	Piping Area Description	Supply and Install(1)
310	Primary Crushing	3.0
311	Ore Storage and Reclaim	10.0
320	Grinding	8.0
330	Flotation and Regrind	16.0
340	Concentrate Handling	30.0
360	Reagents	70.0
390	Tailings Thickening	5.0
420	Water Systems	200.0
430	Sewerage	30.0
470	Air Systems	25.0
500	Ancillaries	25.0
540	Fuel Storage and Distribution	0.0

Note: (1) % based on Area mechanical equipment installed costs.

Installation man-hours and productivity were calculated using a 50% labour / 50% material ratio.

21.4.15.12 Piping for Tailings Delivery and Reclaim Water

Quantities for tailings and reclaim water piping were provided by Ausenco.

Supply pricing and installation unit rates were based on piping material supply by regional suppliers and allowances for installation provided by Merit based on historical in-house data.

21.4.15.13 Electrical Distribution

The electrical estimate was based on single-line diagrams and connected loads, as detailed in the flowsheets. Ausenco and CMMC itemized all major electrical equipment The detailed MTO and electrical equipment list was provided in Excel format.

Ausenco made the following statements:

- Electrical demand based on the mechanical equipment list motor sizes
- Electrical distribution based on site plan
- Electrical equipment design/supply is based on the Load List

- Quantity included as an allowance to cover an amount of the commodity that cannot be derived or calculated.

Budget quotations were obtained by CMMC for the electrical equipment based on preliminary specifications and equipment list provided by Ausenco.

Quantities for in-plant electrical materials were provided by Ausenco. Electrical materials supply costs were provided by CMMC.

Installation unit rates were based on estimated costs supplied by a regional general contractor.

Additional allowances for electrical bulks (cable trays, conduits, etc.) were also included and based on a percentage of the mechanical equipment installed costs. Based on Ausenco in-house historical data, the electrical bulks percentages included for all process areas is shown in Table 21-7.

Table 21-7: Capital Cost Estimate – Electrical Bulks Allowances

Area	Electrical Bulks Area Description	Supply and Install(1)
WBS LVL 2		
310	Primary crushing	5.0
315	Ore storage and reclaim	13.0
320	Grinding	9.5
330	Flotation and re-grind	5.5
340	Concentrate handling (thickening and filtration)	6.0
360	Reagents	17.5
390	Tailings system (thickening)	16.5
420	Water systems	8.0
430	Sewerage	1.5
470	Air systems	5.0

Note: (1) % based on Area mechanical equipment installed costs

Installation man-hours and productivity were calculated using a 35% labour / 65% material ratio.

21.4.15.14 Power Supply

A regional high voltage electrical contractor estimated costs for the incoming high voltage power line and main substation, complete with pricing and labour force schedules.

The 15 km for onsite 11 kV overhead lines were determined from the overall site plan by CMMC. An allowance of $107,000/km as instructed by CCMC was included.

As indicated by CMMC, an allowance was included in the CAPEX for power line access to an existing 220 kV infrastructure located 11 km from site.

21.4.15.15 Instrumentation

Process instrumentation-distribution allowances were based on a percentage of the mechanical equipment supply pricing. Based on Merit in-house historical data, an allowance of 7.5% for instrumentation distribution cost was included for all process areas.

Installation man-hours and productivity were calculated using a 30% labour / 70% material ratio.

The distributed control system (DCS) and communication systems were estimated based on supplier quotes received by CMMC.

Installation man-hours and productivity were calculated using a 10% labour / 90% material ratio for the communication system.

Installation man-hours and productivity were calculated using a 20% labour / 80% material ratio for the DCS.

21.4.15.16 Direct Field Labour

There are an estimated 1.6 million man-hours of direct and indirect construction labour associated with the Project’s construction.

The work rotation will consist of 21-days-on and 7-days-off, 7 d/wk at 10 h/d. Certain contracts, such as mine development and earthworks, may require unique work week hours due to seasonal work conditions.

Surface construction labour rates have been solicited from regional contractors for reference only as the installation unit rates provided by the contractors are all inclusive.

The labour rates include:

- Base labour wage rate
- Union fringe benefits
- Payroll taxes and insurances
- Appropriate composite crew mixes
- Small tools and consumables allowances
- Overtime premiums
- Benefits and burdens
- Workers compensation premiums
- Travel allowances
- Transportation to and from onsite camp accommodations
- Appropriate crew mixes
- Field office overheads
- Home office overheads
- Contractors’ profit.

21.4.16 Project Indirect Costs

21.4.16.1 Engineering and Procurement

Engineering and procurement (EP) costs of $19.3 million were estimated and included for the process plant and surface infrastructure by Ausenco, equivalent to 6.2% of all direct costs, excluding mining and plant equipment, pre-development mining, power supply and TSF costs.

Engineering costs of $2.2 million for the power supply and studies for harmonics and interconnection engineering approvals, as instructed by CMMC were included.

A provision of $1.3 million was included for surveying and miscellaneous concrete and welding testing.

21.4.16.2 Construction Management

Construction management (CM) costs of $15 million was estimated by Merit for the TSF, plant and surface infrastructure, equivalent to 4% of all direct costs, except for mining and plant equipment, and pre-development mining costs themselves. This is based on experience with similar projects.

Site investigations and QA supervision costs of $1.15 million for TSF, as instructed by CMMC were included, equivalent to 4% of all TSF direct costs.

21.4.16.3 Construction Temporarily Facilities and Services

Construction costs not included in the direct costs as unit rates, productivities, material costs, and labour are included in the indirect cost estimate for the Project and include:

- Construction Management field offices, furnishings, equipment
- Temporary power supply
- Temporary water supply
- Temporary hoarding
- Warehouse and lay down costs
- Temporary toilets
- Temporary communications
- On-going and final clean-up
- Yard maintenance
- Janitorial services
- Owner’s site safety; personnel and training.

It is important to note that contractor costs to construct the Project are included in the Direct Costs. Only the costs associated to manage the contracts not included within the contractors’ unit rates are included as an allowance in the Indirect Costs. Unit prices submitted by contractors are “all-in” rates, which include mob and demob, contractor’s construction equipment, operators, insurance, overhead, and profit.

21.4.16.4 Construction Equipment

Construction equipment for major earthmoving, access road, power line, concrete, structural steel, mechanical, plate-work, and electrical disciplines is included in the direct costs since unit prices submitted by contractors are "all-in" rates, which include contractor's construction equipment.

The remainder of the construction equipment costs for factorized piping and instrumentation disciplines were included in the indirect costs mostly using contractor pricing, and in few cases using historical data on similar types of projects to develop an hourly cost for construction equipment per direct man-hour.

21.4.16.5 Construction Accommodation and Catering

An average catering cost of US$36 per camp man-day is based on prices provided by experienced national catering contractors who provided a scale of man day costs based on various levels of camp occupancy. The average considers the higher prices for low camp occupancy and lower costs for high levels, with the average cost calculated using the construction labour force schedule.

21.4.16.6 Freight and Logistics

A freight allowance equaling 5% of the total material cost (when not included in subcontractors' unit rates) and 7.5% to 8.5% of the total mechanical and electrical equipment cost have been included for all procured items. CMMC provided freight estimates on key processing equipment based on actual volumes and weights, which were obtained by an International logistics services provider. When these rates were provided by CMMC the freight allowance factor was not applied.

21.4.16.7 Commissioning and Start-up

An allowance of 1.4% of the total mechanical and electrical equipment costs has been made for retention of vendor representatives for start-up, as well as a selection of twelve people from the contractor's crews and four staff engineers for a period of about 90 days.

An allowance equivalent to 2% of the costs of the mechanical and electrical equipment has been included for start-up spares.

21.4.16.8 Spare Parts

Costs provided by CMMC for an equivalent to 12.3% of the costs for equipment have been included. An allowance of 5% of the electrical equipment has also been included.

21.4.16.9 First Fills

The cost of process first fills was provided by CMMC based on supplier quotations.

21.4.17 Owner's Costs

Owner's costs were developed by CMMC to include the following items:

- Corporate office staff assigned to the Project
- Owner's project management staff

- Owner's home office travel
- Owner's home office general expense
- Owner's field staffing
- Owner's field travel
- Owner's field general expenses
- Recruitment allowance
- Site Security
- Warehousing
- Bussing to and from off-site camps
- Training programs for operations staff
- Builders risk insurance, general liability insurance, political risk insurance, and miscellaneous allowances for deductible claims
- Duties
- Outsourced site services (security and janitorial)
- Project legal costs
- Product marketing
- Land surveys (including roads during construction)
- Metallurgical testwork programs
- Permits and licenses
- Miscellaneous outside consultant
- Right of way and land purchase costs.

21.4.18 Contingency

Contingency was estimated by category, considering items quoted, estimated, or factored. Contingency is subjective and is based on the degree of confidence the Feasibility Study team deems reasonable. It covers items included in the scope of work as described in this Feasibility Study, but which cannot be adequately defined at this time due to lack of more accurate detail. The overall contingency was calculated as approximately 10% of the total costs of Direct and Indirect costs and covers unforeseeable costs within the scope of the design.

Moderate contingencies were applied where the uncertainty in pricing was considered to be relatively low, as a result of firm quotes for equipment and the majority of building being obtained. Higher contingency was assigned to disciplines such as earthworks where a greater level of uncertainty in conditions and quantities is typically expected. Similarly, the allowance for piping was increased to reflect less available detail from engineering, while electrical allowances were reduced commensurate with relatively detailed estimation provided from engineering. The contingencies on Indirect costs and Owner's costs were based on detailed breakdowns of those costs and corresponding detailed estimates of applicable contingency, based on confidence in the level of planning completed.

The quantities provided by the engineers are in the order of what we would expect for a plant of this size, capacity, location, and account for items such as in-ground water, structural loads, and climatic conditions.

Intangible items are in the indirect costs, but are limited to items such as fuel, accommodation, and travel rather than spares, first fills, and similar items.

21.5 Operating Cost Estimation

21.5.1 Background

The mining strategy adopted for this Feasibility Study includes the Owner's owning of the major mining equipment with the Owner responsible for operating and maintaining all equipment to achieve a consistent mill feed rate of up to 31,200 t/d for the life of the Project. Mining costs associated with mobilization, clearing and grubbing, earthworks projects, mine pre-stripping activities, and the Owner's management prior to commissioning of the process plant have been capitalized.

As a part of this Feasibility Study, CMMC updated the operating cost estimate for the processing plant. Operating cost adjustments were made for the secondary crusher and HPGR format versus a semi-autogenous grinding (SAG) mill circuit based on recent vendor quotes for consumables. Processing plant labour was increased to support TMF construction and management.

21.5.1.1 Mine Operating Cost Estimation

Provisions for all operating costs associated with blasting activities for the Little Eva and Satellite pits were made. Mining sequences, schedules, and infrastructure layouts were generated by CMMC for ore and waste haul routes, and to calculate operating costs for the Project.

CMMC reviewed and validated the mining costs. In generating their estimate, CMMC made the following assumptions:

- The loading machinery was selected to meet the production schedule, orebody characteristics, grade control accuracy, and operating on a continuous basis (24 h/d, seven d/wk).
- Mobile equipment operating costs include maintenance wage requirements, oils and lubricants, replacement of major maintenance components, daily servicing, tires/tracks, and ground engaging tools.
- Service intervals and major component replacement schedules were based on operating experience, and original equipment manufacturer recommendations.
- Equipment fuel burn estimates were calculated by CMMC based on current operating performance or original equipment manufacturer specifications as appropriate.
- Current labour rates were used.
- Ancillary plant (e.g., dozer, grader, water cart, etc.) hours were calculated as a ratio of loading unit hours, and in line with current operating practice.
- Equipment ownership costs assuming all new equipment, zero residual value.

21.5.1.2 Processing Plant and Infrastructure Cost Estimation

The processing plant operating estimate has been generated by CMMC with consideration of:

- Consumable usage rates determined by metallurgical testwork
- Power consumption based upon the established equipment drive list and usage factors

- Assumption of plant operating availability of 92% giving 8,059 h/a operation
- Initial power cost of $0.1211/kWh for the first three years of operation, followed by $0.0635/kWh for the remainder of the Project life, based on the completion of the CopperString transmission line to the East coast.
- Salaries were reviewed and based on the McDonald Gold and General Mining Industry Remuneration Report (Australasia).

21.5.2 Mining Operating Cost

21.5.2.1 General

Mining is based upon a continuous 24/7 mining operation at a rate designed to deliver a maximum of 11.4 Mt/a of ore to the mill and consists of a total movement of 584 Mt of material including re-handled material.

Four material movement cycles will be in operation:

- Ore for processing plant feed will be transported to the run-of-mine (ROM) pad for direct dumping of sulphide ore into the crusher feed bin
- Native copper ore will be stockpiled adjacent to the ROM pad and fed by frontend loader, not to exceed 25% of total feed
- Low-grade ore will be stockpiled adjacent to the ROM pad, or on top of a waste dump for future processing later in the mine life
- Waste material will be transported to the location for construction of pit bunds, other required infrastructure, or directly to waste dumps adjacent to each relevant pit.

CMMC used typical travel speeds and duty cycles for the selected truck types, road conditions, and payloads developed truck haul cycle times, which were based on the current layouts and haul profiles. Provision of diesel fuel for mining activities is priced at US$0.50/L after rebates. Typical tire costs used for the haul trucks is approximately $10,200 per tire.

21.5.2.2 Scope

To maximize cost effectiveness, the Owner intends to provide the fuel and explosives to the fleet, which CMMC will also operate and maintain.

Accordingly, it is intended that the Owner meets part or all the lease and financing cost of the fleet, if this option is selected.

Accordingly, the mining operating costs have been based on the following division of scope within the mining area. Owner's scope:

- Supply of diesel fuel
- Commute flights, accommodation and messing for all mining personnel
- Supply of explosives and "down the hole" services
- Selection of mining equipment
- Equipment operation with CMMC operators

- Maintenance facilities and servicing of equipment including spares, tires, and consumables provision
- Provision of minor equipment and tools
- Offices and provision of water, communications, and electrical services from the nominated tee-off points.

21.5.2.3 Mining Cost Estimate

Table 21-8 shows the LOM, estimated unit mining costs based on the given mining schedule. The exchange rate used was 1.55 (AU$:US$). Equipment, supplies, repair and maintenance, and personnel labour costs were generated for the calculated mining fleet based on the mining schedule (R12-CMMC internal designation for revision number).

For completeness, the data includes all pre-stripping costs incurred prior to commissioning and subsequent mining pushbacks during operations.

Note that:

- Costs associated with loading and long-distance truck haulage of ore from the Ivy Ann satellite pit to the ROM pad are included in the mining cost operating estimate.
- Provision for re-feed of 100% of stockpiled ore in addition to a proportion of the direct feed ore to the crusher has been made in the processing operating costs.

Additionally, there is provision for four, sampling, pit technicians within the mining general and administrative (G&A) to cover sample collection, bench mark-up, and load control.

Table 21-8: Mining LOM Operating Costs

Operating Cost by Category	LOM Average Cost Total (US$/t)	% of Total
Hauling	0.60	36
Loading	0.21	13
Road maintenance	0.15	9
Pit cleanup and maintenance	0.02	1
Dump maintenance	0.08	5
Drilling	0.21	13
Blasting	0.17	10
Support	0.03	2
Stockpile	0.01	1
Mine Contractor-IA	0.04	2
Dewatering(1)	0.02	1
G&A	0.11	7
LOM Average Cost Total	**1.66**	**100**

Note: (1)Dewatering labour accounted for in processing operating cost totals.

The LOM operating and predevelopment capital costs are summarized in Table 21-9. For details on calculation of capitalized and operating pre-stripping, refer to Section 22.

Table 21-9: Mine Production Capital and Operating Costs

Production Mining Cost Category (Year 1 to Year 15)	Cost ($ million)	Ore ($/t)	Mined ($/t)	Moved ($/t)	Pound ($/lb)
LOM Total Mining Operating Cost(1)	888.7	5.25	1.66	1.56	0.60
Direct Capital Cost(2)	71.8	0.42	0.13	0.13	0.05
Sustaining Capital Cost(2)	34.0	0.20	0.06	0.06	0.02
Mining and Process Material Movement (Year 1 to Year 15)	**Unit**	**Value**			
Total Ore Mined	(t '000s)	169,217			
Total Waste Mined	(t '000s)	367,054			
Total Material Mined	(t '000s)	536,271			
Total Re-handle Moved	(t '000s)	31,833			
Total Material Moved	(t '000s)	568,104			
Total Tonnes Processed	(t '000s)	169,217			
Total Copper Produced	(Mlb)	1,485			

Notes: (1)Total does not include pre-development cost and material movement. (2)Includes $400,000 in Year -1 of sustaining cost.

21.5.2.4 Mining General and Administration Costs

The mining operations G&A consists of salaries and associated on-costs for CMMC's management, costs for camp accommodation and flights that will be free issued to the mining personnel and miscellaneous expenses as summarized in Table 21-9.

21.5.2.5 Owner's Personnel

Provision for personnel wage costs have been made based upon wage information from the April 2019 McDonald & Company remuneration survey for the mining and resources industry. The Owner's employees will be responsible for the following functions:

- Mine management
- Resource definition
- Geological and grade control
- Mine planning
- Survey
- Dewatering
- Mine operations and supervision
- Mine maintenance and supervision.

Employee additional costs (on-costs) were split into three main categories:

- Employment on costs

- Accommodation at a rate of US$36/man-day
- Flights at a rate of US$420/roundtrip (rotation).

Employment on-costs were incorporated to account for the following:

- Superannuation
- Leave allowance
- Government payroll tax/levies
- Workers compensation payments
- Staff amenities.

21.5.2.6 Personnel Costs

The Owner will provide camp services (inclusive of accommodation) and flights for the mining personnel, and most of the personnel will be working on a 7:7 roster. The estimate has been generated based on annualized camp man-day and flight requirement for each of the Owner's personnel at a rate of $36 per man-day and $420 per roundtrip (rostered return) flight.

21.5.3 Processing Plant Operating Cost

21.5.3.1 General

The operating cost estimate for the processing plant has been updated for the treatment of 11.4 Mt/a of ore from the Little Eva deposit and satellite deposits and has been compiled by CMMC with the assistance of Ausenco from a variety of sources including:

- First principle estimates
- Supplier quotations
- CMMC advice
- Metallurgical testwork results
- Ausenco project standards.

Power draws on comminution equipment were estimated using the 70th percentile design criteria for or competency and hardness as outlined in the plant design. The financial model assumes two separate power unit costs based on the completion of the copper string project. See Section 18 for more information.

The operating cost estimate for the processing plant inclusive of fixed and variable costs has been developed using the plant parameters summarized in Table 21-10.

Table 21-10: Plant Operating Conditions

Item	Unit	Average Values
Mill feed rate	t/h	1,413
Operating days	d/a	365
Plant utilization	% of total time	92
Operating hours	h/a	8,059
Tonnes	t/a	11,388,000
Head Grade	% Cu	0.46
Recovery	%	87
Concentrate grade	% Cu	28

21.5.3.2 Scope

The scope associated with the plant and infrastructure operating estimate includes all costs for operating each process unit within the plant up to the point of production of concentrate:

- Process plant operations personnel
- Maintenance personnel
- Contract maintenance requirements
- Grinding media, HPGR and crusher wears, consumables, reagents, and spares
- Oils, lubricants, and fuels
- Grid power
- Laboratory services
- Vehicle operating costs
- Heavy equipment operating costs for TSF maintenance
- Administration services.

21.5.4 Clarifications and Exclusions

The following items have been excluded from the estimate of operating cost for the processing plant:

- All head office costs and corporate overheads
- Concentrate treatment and refining costs (provision in financial model)
- Maintenance of all mine and plant access roads (in mining estimate)
- Royalties (provision in financial model)
- Exchange rate variations
- Escalations
- Project financing costs
- Interest charges
- Land compensation costs (traditional landowners)

- Subsidies to local community
- Contingency
- Plant site rehabilitation costs (provision in financial model)
- Starter dam construction
- Rockfill dam raises.

Items of note:

GST has not been applied, as income generated will be GST free because the copper concentrate product is considered an export commodity and will not attract output credits. The GST has cash flow implications, as it will apply to most inputs including consumables. GST paid on these items can be claimed back from the Australian Tax Office.

21.6 Owner's Personnel

The mine and plant will operate continuously. The labour estimate is based on two 12-h shifts each day and a two-weeks-in/two-weeks-off roster with a fly-in fly-out (FIFO) arrangement and on-site camp accommodation. Labour and on-costs for the processing plant (excluding specialist contract labour) is $0.84/t milled and represents most of the process plants fixed cost of $1.57/t.

Table 21-11 shows the manning schedule for the processing plant, which allows for a total of 93 personnel to manage, operate, and maintain the facility and associated infrastructure. There will be four shift operating crews comprising a shift supervisor and eight operators. Two operators and three metallurgical technicians will normally workday shift unless relieving shift operators who are on leave. Maintenance personnel will workday shifts with a call out roster for night shift.

A daily rate of $36 per person has been used for accommodation and meals. An allowance of $420 has been used for flights to and from site.

Table 21-11: Manning Schedule for the Processing Plant

Position	Number
Production	
Plant Manager	1
Production Superintendent	1
Senior Metallurgist	1
Senior Chemist	1
Plant Clerk	1
Plant Metallurgist	3
Metallurgical Technicians	1
Assayers	2
Sample Preparation Technicians	6
Operations Shift Supervisor	4
Tier 1 Process Technician	8
Tier 2 Process Technician	12
Tier 3 Process Technician	12
General Day Labourer	4

Position	Number
Maintenance	
Maintenance Superintendent	1
Maintenance Planner	2
Mechanical Supervisor	2
Fitter	12
Mechanical Apprentice	4
Lubrication Serviceman	2
Boilermaker	4
Electrical Supervisor	2
Electricians	4
Instrumentation Technicians	2
Electrical Apprentice	1
Total	**93**

21.7 Consumables Costs

Table 21-12 shows a summary of operating consumable unit consumption (kg/t mill feed) and costs ($/kg) for grinding media and reagents.

Reagent usage rates have been based on abrasion rates and has been estimated based on power consumption and benchmarked with operations treating similar ore.

Concave liner life has been estimated equivalent to one complete set per year including spider cap and spider and pinion arm liners. Primary crusher mantles are expected to be consumed at a rate of two sets per year. Seven sets of secondary crusher bowl and mantle sets are assumed. HPGR tire assemblies are estimated to have 11,000 hours of life based on vendor feedback and pilot testwork. The cost estimate for ball mill liners is based on a complete lining change each year. Liner costs associated with the regrind Vertimill® are mainly for wear linings of the screw and were supplied by Metso.

Table 21-12: Operating Consumable Consumptions and Costs (±10%)

Item	Consumption (kg/t)	Unit Price ($/kg)	Cost per Tonne of Plant Feed ($/t) (31,200 t/d)
Grinding media – 52 mm	0.500	1.068	0.53
Grinding media – 16 mm	0.054	1.381	0.07
Collector – PAX	0.090	2.35	0.21
Frother – H27	0.030	2.46	0.07
Flocculant (concentrate and tailings)	0.080	3.25	0.19
Sulphidizer – NaHS	0.400	1.15	0.46
Total			**1.55**

21.8 Maintenance Costs

Maintenance material costs and specialist contract labour costs have been estimated from data base information and vendor spare parts lists. These costs (excluding direct labour costs) are approximately 4.5% of the installed equipment CAPEX.

21.9 Power Costs

Electrical power will initially be supplied from the North Queensland grid at an anticipated unit cost rate of $0.1211/kWh. Following completion of the copper string project, allowing for power transmission from the East coast of Queensland, this power cost will drop to $0.0635/kWh, as per the current agreement. The average power draw for each drive has been calculated from the installed power and application of utilization and efficiency factors depending on the duty. Comminution equipment power draws have been estimated using Ausgrind and the 70th percentile design case for ore competency and hardness.

21.10 Processing Operating Costs

The processing plant operating cost estimate is presented below in summary. The estimate is considered to have an accuracy of ±10%, is presented in United States dollars (US$) and is based on prices obtained during Q4 2019 and Q1 2020.

The operating cost estimate by plant area is summarized in Table 21-13.

Table 21-13: Plant Operating Cost Summary 31,200 t/d (±10%, Average LOM Values)

Description	Total Cost ($ '000s)	Unit Cost ($/t milled)
Labour (includes maintenance labour)	7,840	0.69
Electric Power	19,744	1.73
HPGR tires, grinding media, and liners	8,933	0.78
Reagents	10,684	0.94
Other process consumables	2,832	0.25
Maintenance	8,483	0.74
Total Process Operating Costs	**58,517**	**5.14**
Milled tonnes (Mt/a)	11,388	
Milled tonnes (t/d)	31,200	

Note: At $0.1211/kWh (Year 1-3); $0.0635/kWh (Year 4, onwards)

21.11 Site Administration Costs

Site administrative costs are expenses not directly related to the production of copper concentrates and include matters not directly related to mining, processing, refining, and transportation costs.

The site administration costs for the Eva Copper Project are determined for 15 years of operations with an average cost of $0.56/t milled. Site administration costs include:

- Administration personnel
- Safety equipment and clothing
- Medical, IT, and training
- Vehicles, buildings, and property taxes
- Communications
- Insurance, consultants, and sanitation
- Community relations and permitting.

The site administration costs over the LOM are estimated at $95 million with an annual cost of $6.3 million.

Transportation costs were estimated based on having a main workforce rally point located in Brisbane. Pricing on flights was based on charter estimates which support the selected hourly roster of 7-days-on, 7-days-off. Camp costs were estimated based on contract management and services within the camp facility. With the proximity of Cloncurry to the Project site, an assumption was made that 25% of the workforce would choose to live locally. The respective camp and flight costs for this portion of the workforce was deducted from the transportation estimate, however, this was offset by a 15% base salary uplift for the local workers.

21.12 Summary of Operating Cost Estimate

The operating cost estimate has been generated as listed below, and operating costs are summarized in Table 21-14 using outputs from the financial model:

- Mining operations
- Processing plant operations
- G&A
- Transport and logistics
- Treatment and refining
- Royalty Costs.

Table 21-14: Operating Cost Estimate – Summary by Area

Operating Cost Area	LOM Total ($ million)	Unit Cost ($/t milled)
Mining	888.7	5.26
Processing	868.3	5.14
G&A	95.0	0.56
Accommodation and Travel	72.4	0.43
Total	**1,924.5**	**11.39**

Notes: Total mining costs are estimated at $5.26/t milled, or $1.66/t mined. Royalties for LOM total is $199.9 million at a unit cost of $1.18/t milled.

22 ECONOMIC ANALYSIS

The Eva Copper Project has been valued using a discounted cash flow approach in the determination of the net present value (NPV), payback period, and internal rate of return (IRR) for the Project. Annual cash flow projections were estimated over the life of the mine based on the estimates of capital expenditures, production costs, and sales revenue. Sales revenue is based on the production of copper concentrate with gold credits.

The Economic analysis includes the entire project life, comprising two years of detailed engineering and construction and 15 years of mining and milling. The valuation date on which the NPV and IRR are measured is the commencement of construction in Year -2.

Sensitivity analyses were performed for variations in copper price, copper grade, copper recovery, operating costs, capital costs, and exchange rates to determine their relative importance as Project value drivers.

This Technical Report contains forward-looking information regarding projected mine production rates, construction schedules, and forecasts of resulting cash flows as part of this study. The mill head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

The estimates of capital and operating costs have been developed specifically for this Project and are summarized in Section 21. They are presented in 2020 United States dollars (US$). The economic analysis has been run with no inflation on a constant dollar basis

22.1 Assumptions

The economic model was created using various assumptions that are based on current and projected future expected economic conditions including, but not limited to, sales prices, operating costs, annual production, ore grades, and exchange rates.

Table 22-1 outlines the key inputs and assumptions used.

Table 22-1: Key Inputs and Assumptions

Parameter	Unit	Value
Mine Life	years	15
Total Ore	Mt	170
Total Waste (including 14,074 kt of oxide material)	Mt	381
Processing Rate	kt/d	31.2
Average Cu Head Grade	%	0.46
Cu Recoveries	%	87
Au Recoveries	%	78
Cu Produced	Mlb	1,502

Parameter	Unit	Value
Au Produced	koz	205
Cu Price (long-term from Year 2)	US$/lb	3.04
Au Price (long-term from Year 2)	US$/oz	1,362
Exchange Rate	AU$:US$	1.55

Other key inputs and economic factors include the following:

- Discount rate of 8% (sensitivities of other discount rates have been calculated).
- Revenues, costs, and taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment.
- Progressive reclamation totalling to $14 million over the LOM.
- Nominal 2020 dollars with no inflation and on a constant dollar basis.
- Results are presented on a 100% basis; do not include management fees. The Capital cost of $49.6 million of mine equipment purchased in Year -1 and Year 1 has been amortized over a lease term of seven years at 5%.
- All pre-development and sunk costs, such as exploration and resource definition costs, engineering fieldwork and studies costs, and environmental baseline studies, were excluded. However, pre-development and sunk costs are utilized in the tax calculations.

22.2 Sales and Net Smelter Return Parameters

Mine revenue will be derived from the sale of copper concentrates averaging 28% Cu and 3 g/dmt Au. The material will be considered a “clean concentrate” with no deleterious elements that cause smelters to penalize the material. A contractual arrangement exists between the Project and Glencore International AG for all of the mines output for the first five years of the Project commencing with the start of mine production. The sale of the concentrate will be made on the basis as freight carrier at (FCA) Seller’s mine gate and based on the annual prevailing market terms (annual benchmark). Details regarding the terms can be found in Section 19.

Marketing cost assumptions used for the economic model are based on discussions with major smelters and concentrate trading companies and on the Company’s own views and experience in the copper concentrate market. For copper-gold concentrates it assumed that LOM treatment charges and refining charges (TC/RC) will average $76/t, $0.076/lb, respectively, and that no penalties will be payable. Copper payability is assumed to be 96% and gold approximately 91%. Copper concentrate production and sale is assumed to begin during commissioning in Q4 Year -1 and continue for 15 years.

Figure 22-1 and Figure 22-2 show the production profile over the Project’s 15-year life.

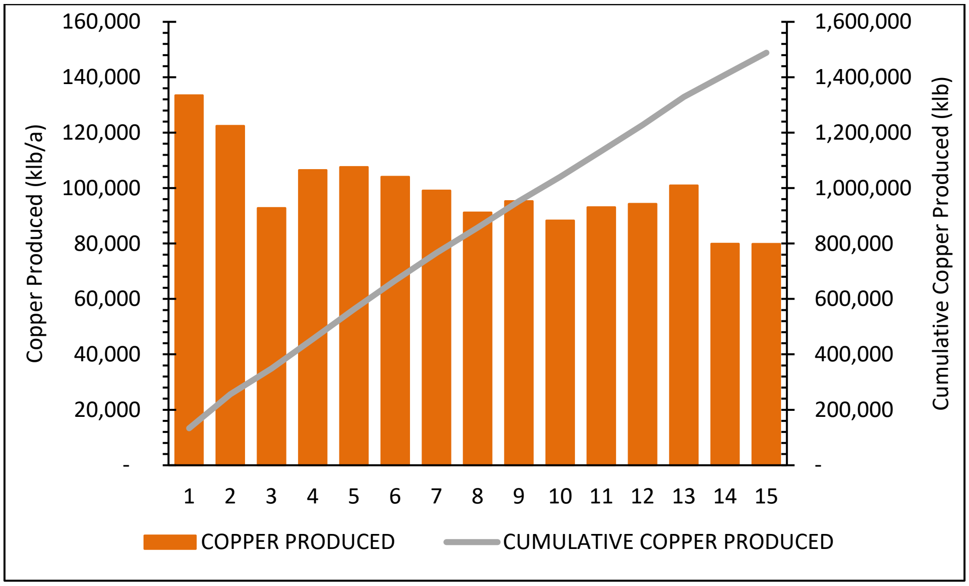


Figure 22-1: Eva Copper Project LOM Copper Production Profile

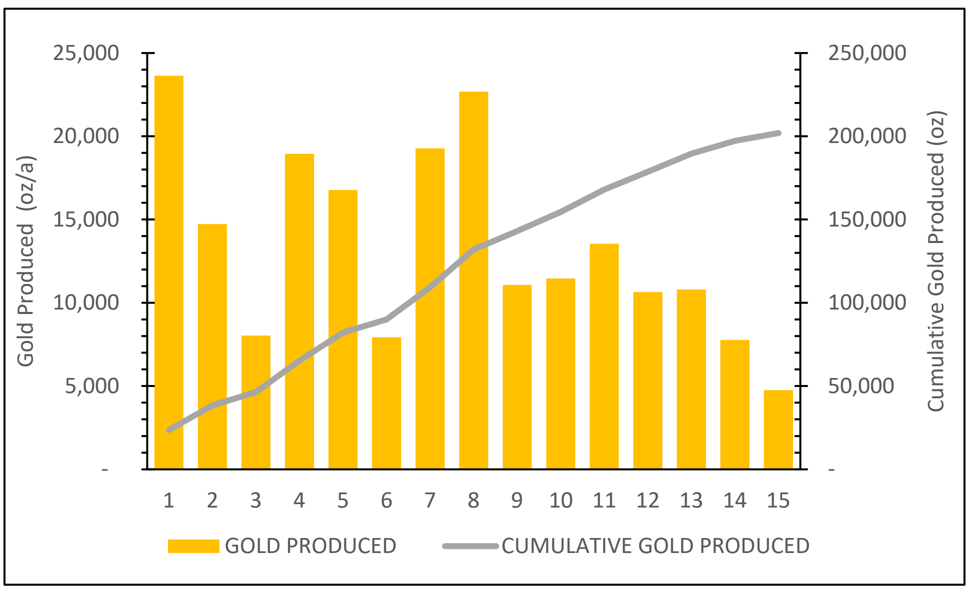


Figure 22-2: Eva Copper Project LOM Gold Production Profile

22.3 Taxes

The Project is subject to Australian corporate tax, which has been applied at 30%. Tax calculations are impacted by depreciation deductions for capital items and other applicable deductions. As at the date of completion of this feasibility study, the Company had approximately $56.7 million in tax losses available. These tax losses have been included in the economic analysis of the Project.

22.4 Royalties

Third party royalties have been considered in this economic analysis and are payable to several separate parties including the State of Queensland.

The economic analysis for the Project accounts for the following royalties at various rates and locations within the Eva Copper Project:

- Queensland Royalty is a modified net smelter return (NSR) royalty on copper and gold at a variable rate between 2.5% and 5.0% depending on average metal prices
- Pasminco Royalty is an NSR royalty of 1.5% payable to MMG (85%) and Lake Gold (15%)
- PanAust Royalty is an NSR royalty of 1.6% or 1.1%
- Dominion Royalty is an NSR royalty of 0.4% or 0.9%
- Kalkadoon People is an NSR royalty of 0.22%.

Royalties of approximately $200 million are payable over the life of the mine to the Queensland government and the above noted private entities. Table 22-4 provides a summary of the LOM taxes, royalties, and other government fees for the Eva Copper Project. Details related to third party royalties are outlined in Section 19.

22.5 Economic Results

The Project is economically viable with an after-tax IRR of 29% and NPV at 8% of $437 million with a 2.5 years payback. Figure 22-3 shows the projected cash flows from the economic analysis and Table 22-2 summarizes the detailed results of this evaluation.

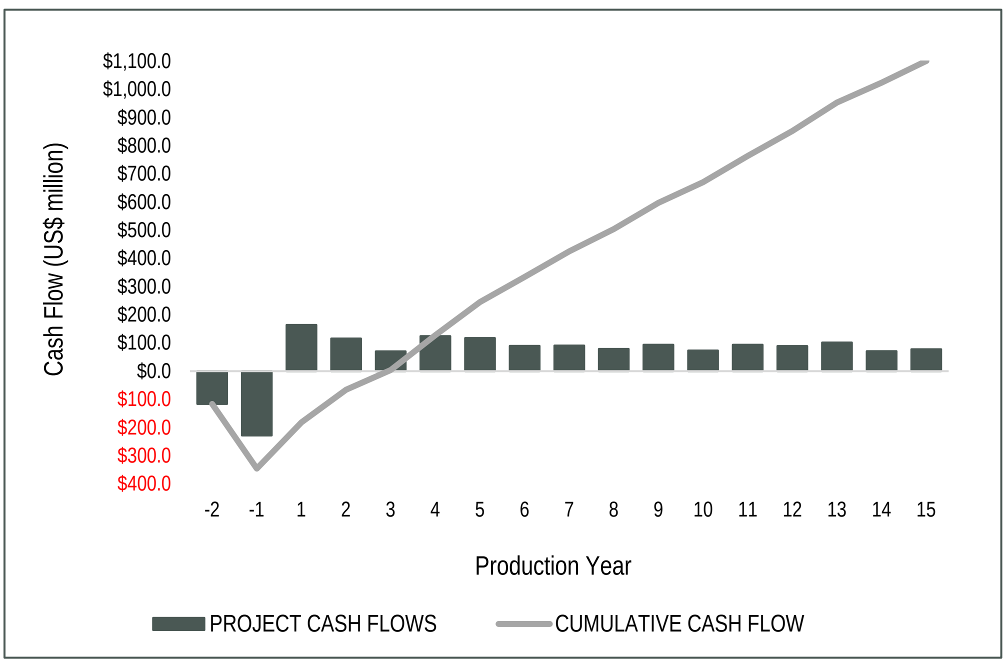


Figure 22-3: Eva Copper Project Annual and Cumulative After-Tax Cash Flows

Table 22-2: Summary of Economic Results

Key Financial Metrics	Unit	Value
Net Revenues	$M	4,311
Operating costs	$M	1,925
Cash flow from operations	$M	2,386
Royalties and Transportation	$M	371
Taxes	$M	447
Cash flow after taxes	$M	1,568
Sustaining capital costs	$M	34
Cash flow after taxes and sustaining capital	$M	1,534
C1 cash cost per pound of copper produced after credits	$/lb	1.44
Cash cost per pound produced (after taxes and sustaining capital)	$/lb	1.76
Pre-Tax NPV 8%	$M	648
Pre-Tax IRR	%	37
After-Tax NPV 8%	$M	437
After-Tax IRR	%	29

Table 22-3: LOM Cash Flow

	Unit	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Revenues																			
Revenues	$M	4,311	-	49	395	353	364	315	315	294	295	278	274	255	271	271	289	228	224
Less expenditures																			
Capital	$M	443	117	230	35	29	7	7	8	8	3	-	-	-	-	-	-	-	-
Operating	$M	1,925	-	35	128	149	156	125	134	147	147	148	120	128	117	118	118	103	88
Sustaining	$M	34	-	-	1	1	2	2	2	2	2	2	2	3	3	3	3	3	3
Royalties & Transportation	$M	371	-	-	28	30	27	25	25	29	26	26	25	22	20	24	24	22	19
Taxes	$M	447	-	-	39	28	2	32	32	23	26	23	34	29	38	37	43	30	31
Total	$M	1,091	-117	-231	165	116	70	124	115	86	91	79	94	73	93	89	101	70	74

The Project will generate an average annual net operating surplus of $190 million in the first five years of full production (revenues net of TC/RCs). The average operating cost is $11.39/t compared with average revenue of $25.52/t milled.

22.6 Sensitivity Analysis

A sensitivity analysis was performed to test Project value drivers on the NPV using an 8% discount rate. Sensitivities to copper price, head grade, copper recoveries, operating costs, capital costs, and exchange rate (AU$:US$) were conducted by adjusting each variable up and down by as much as 20% independently. As with many metal mining projects, the results of the analysis revealed the Project is most sensitive to copper price, copper recovery, and copper head grades.

Table 22-4 and Figure 22-4 show the results of the sensitivity testing as run through the economic model.

Table 22-4: Cash Flow Sensitivities

	After-Tax NPV (8%) ($ million)				
Variable	**0.80**	**0.90**	**1.00**	**1.10**	**1.20**
Cu Price	137	286	437	587	737
Cu Recovery	156	297	437	577	717
Cu Grade	166	302	437	572	707
Exchange Rate	348	396	437	470	498
Capital Cost	495	466	437	408	379
Operating Cost	576	506	437	367	298

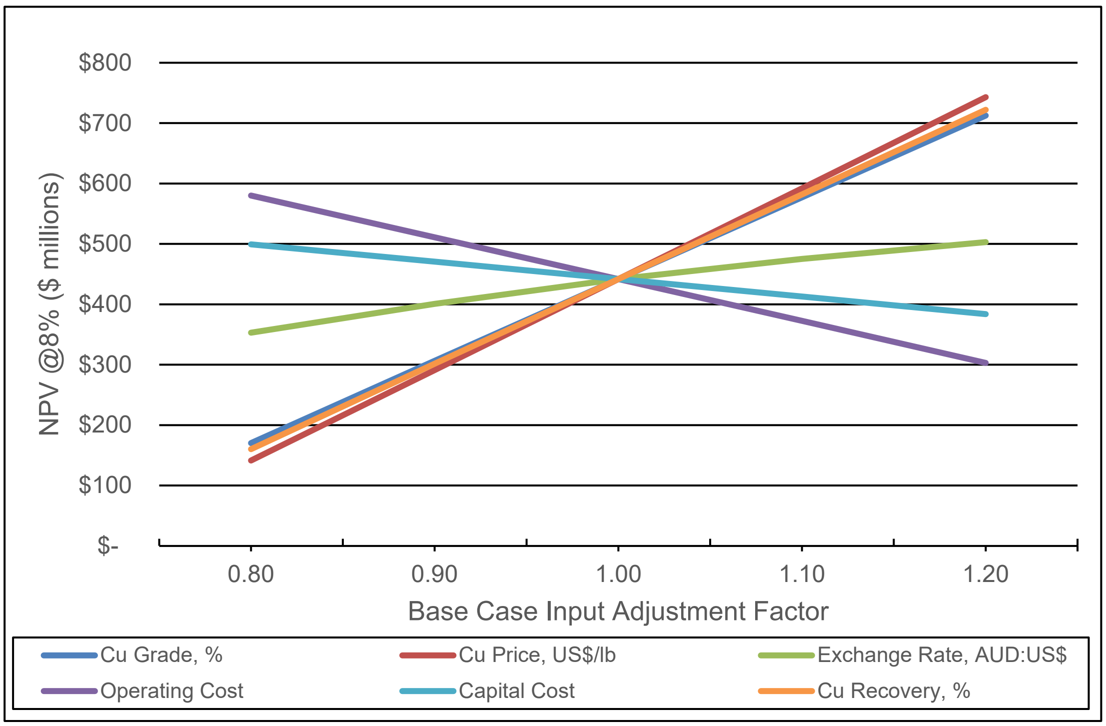


Figure 22-4: Eva Copper Project Sensitivity Analysis

23 ADJACENT PROPERTIES

23.1 Mining Properties (Regional)

Mount Isa was established on the discovery of world-scale copper-zinc-lead deposits in 1923. A major mining complex and a city of 22,000 people have grown on the site in the last 94 years, with multiple open pit and underground mines, smelters, mills and flotation plants, and a sulphuric acid plant. The town hosts many mining suppliers and service organizations, and has a deep pool of skilled mining industry people. Mt. Isa has two electric power generators supplied by a natural gas pipeline from South Australia, an airport, rail, and other services.

Cloncurry was established much earlier than Mount Isa, in 1867, on the discovery of copper by Ernest Henry, and the town was founded in 1884.

There are numerous active mines in the area. In addition to Mount Isa, there are five major active mines: Ernest Henry copper-gold mine and Lady Loretta lead-zinc-silver mine, both owned by Glencore; Cannington silver-lead mine, owned by South 32; the Dugald River zinc-lead-silver mine, owned by MMG; and the Mount Gordon copper-gold mine, owned by Capricorn Copper. All are major, internationally important mines.

Smaller operations (active or in care and maintenance) include Osborne copper-gold mine, owned by Chinova; Mount Colin copper mine, owned by Round Oak Minerals, Lady Annie copper-gold mine, owned by CST Mining; Mount Cuthbert Copper mine, owned by Malaco Mining; Rocklands copper-gold mine, owned by Cudeco; and Eloise copper-gold mine, owned by FMR Investments.

The only major closed mine is the Mary Kathleen Uranium mine.

23.2 Mining Properties (Adjacent)

Mining properties that surround the Eva Project are predominantly Exploration Permit for Minerals (EPM) held by the CMMPL (Figure 23-1). These properties cover a highly prospective north–south corridor with similar geology to that which hosts the Project's Mineral Resources, where numerous copper-gold mineralized prospects have been established and are being systematically explored. No additional Mineral Resources have as yet been defined.

The major Dugald River zinc-lead-silver mine owned by MMG is located 11 km south of the planned Eva Copper Project mine site, within a Mining Leases (ML) surrounded by MLs and EPMs held by the Company. The mine was commissioned in November 2017. MMG indicates that the mine will process an average 1.7 Mt/a of ore, to initially produce 170,000 tonnes of zinc concentrate, plus byproducts. The mine will operate over an estimated 25 years while the ore body remains open at depth. The mine is an underground operation accessed via declines. Published Measured, Indicated, and Inferred Mineral Resources are: zinc resources of 64.8 Mt at 12% Zn, 2.2% Pb, and 31 g/t Ag (plus stockpiles of 0.23 Mt at 10.8% Zn, 1.7 Pb, and 49 g/t); and copper resources of 4.4 Mt at 1.8% Cu and 0.2 g/t Au. Published Proven and Probable Ore Reserves are 32.8 Mt at 11.9% Zn, 2.2% Pb, and 44 g/t Ag. Resources and Reserves are from MMG 2017 statements published in accordance with Joint Ore Reserves Code (JORC) 2012 edition (JORC, 2012). Stratigraphy interpreted to be prospective for similar zinc mineralization is identified within the tenure held by the Company surrounding the Dugald River Project.

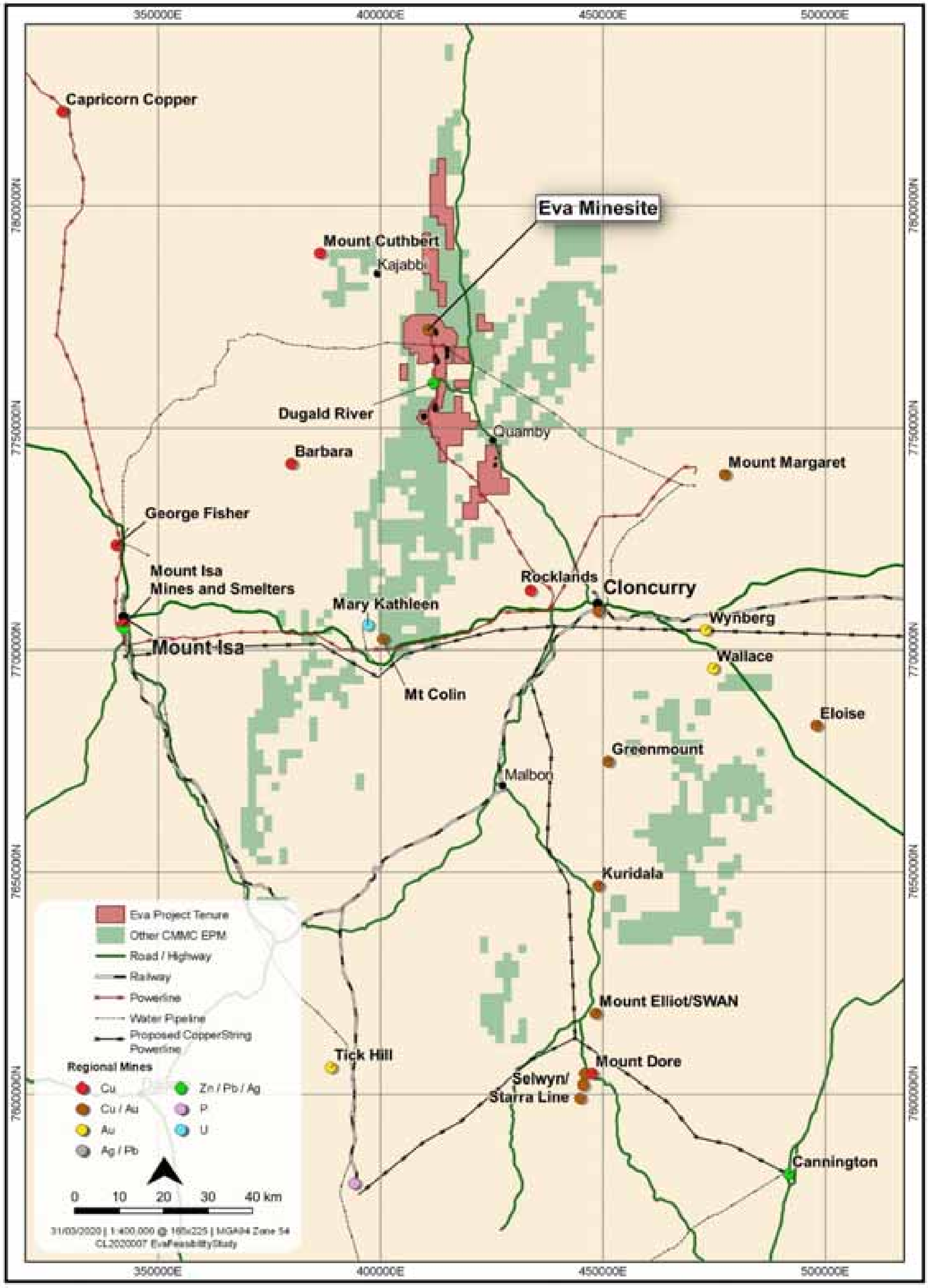


Figure 23-1: Adjacent Mining Properties and Major Mines around the Eva Copper Project

23.3 Non-Mining Properties

Immediate key local non-mining stakeholders associated with the Eva Copper Project are landowners, leaseholders, the Kalkadoon people, and state and local governments. They are:

- Landowner: Harold MacMillan (Mt. Roseby Homestead)
- Landowner: North Australian Pastoral Company (Coolullah Homestead)
- Kalkadoon people
- Commonwealth and Queensland State departments
- Cloncurry Shire Council.

CMMPL has been in continuous communication with the above stakeholders for many years. Refer to Section 4.4 regarding Pastoral Leases and Compensation Agreements with the four pastoral landholders for both the MLs and key areas of activity in the surrounding EPMs.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Plan Outline

24.1.1 Plan Objective

The development of a practical Project Execution Plan (PEP) outline at this stage of the Project is integral to the success of the next phase of the work as it enters the basic and detailed engineering stage. The PEP should be completed at the commencement of the next stage of project implementation, and helps form the basis for the ongoing work insofar as it provides the blueprint upon which assumptions were made during the feasibility study and the cost and schedule conclusions reached at the end of the study. Key assumptions were based on experience about how the Project would be developed in its location, in the time frame planned, with an operational timeline developed through the feasibility planning process.

In general, the PEP provides a platform to support the successful interaction of the engineering, procurement, and construction activities. This interaction embraces communications, technical and practical issues, safety, environmental, governmental, social issues, and all other facets of the Project that end when the operation starts and the Owner takes control of plant operations.

The PEP guides the reader through Project development generalities, describing how the Feasibility Study Team assumed the Project should move forward through to Mechanical Completion, after which the Owner will take control of plant operations. The PEP includes discussion of the following activities to be managed:

- Basic and Detailed engineering
- Procurement of long-lead delivery equipment
- Freight
- Communication systems and information management
- Project controls
- Project schedule
- Construction field requirements
- Ordering bulk materials
- Site environmental requirements
- Site safety requirements
- Site security requirements
- Construction resources
- Accommodation for construction and operating workforce
- Commissioning the plant and handover to Owner.

24.1.2 Plan Execution Strategy

24.1.2.1 Pre-Construction / Project Setup

The pre-construction phase includes activities during the period leading up to the financing and permitting approvals. Continued development could encompass the following:

- Additional geotechnical drilling and test pit drilling to support final design
- Continuation of environmental monitoring
- Project design and construction optimization
- Researching local resource availability
- Negotiation with long-lead delivery vendors
- Establishing the availability and suitability of contractors
- Preparing the Project management administration office at the Project laydown area
- Finalizing Owner commitments to the Project, including the mine plan
- Improving site access.

24.1.2.2 Basic Engineering Phase

The overall goals for the Basic Engineering phase are:

- Develop the Project management control documents and baselines, including the capital cost control estimate and master schedule
- Update the engineering design to incorporate any further process improvements desired by Copper Mountain Mining Corp. (CMMC)
- Finalize the site water management design
- Continue to commit to long-lead delivery equipment and contracts
- Prepare RFQ packages for schedule-critical equipment to be ready to issue for tender
- Prepare approved-for-construction status engineering for the early works, including earthworks such as access and plant roads
- Prepare approved-for-construction status engineering for the tailings storage facility (TSF) and site water management structures.

The Basic Engineering phase will be utilized to optimize the PEP, including integration of the overall Project schedule.

24.1.2.3 Detailed Engineering Phase

The tasks that are required in advance of construction activities include, but are not limited to, the following:

- Develop the Project management control document
- Finalize flowsheets
- Order long-lead delivery equipment: crusher, main transformers, and large motors and pumps
- Complete General Arrangement drawings

- Complete site water balance
- Conduct constructability reviews
- Advance bulk earthwork drawings for construction
- Finalize tailings dam construction stage drawings
- Design Power transmission line route
- Survey overhead power transmission line
- Source suitable site aggregate and process tenders for early work activities
- Mobilize construction management (CM) personnel for early work support
- Arrange for the operations and construction camp units' relocation and catering services
- Establish boiler plate for contracts and purchase orders
- Finalize the Project schedule based on all the information gathered to that point
- Establish the cost reporting and control system
- Establish the field survey contract
- Establish the quality assurance (QA) contract
- Arrange the freight forwarding contract
- Arrange the temporary construction facilities, including fuel and water.

24.1.2.4 Procurement

The Owner's Project Engineering and Procurement (EP) team is in place to ensure that the EP activities will support the early works program. The early works program consists of securing critical path long-lead equipment items and establishing the bulk earthworks and civil programs required to support the construction schedule.

The EP team will develop packages that will support the contracting strategy for procuring concrete, buildings, structural steel, and piping. Work packages will be bid and awarded continuously based on availability of engineering information.

24.1.2.5 Construction Phase

Although the Mining Leases (ML) and Environmental Authority (EA) permits are in place, construction activities cannot start until a Major Amendment to the EA and a Progressive Rehabilitation and Closure Plan (PRCP) are lodged and approved and Financial Assurance (FA) commitments have been met. The lead time for these is up to six months, and they are therefore not considered to present a major constraint in the schedule (see Section 24.2). Other secondary state approvals and permits are required on completion of detailed engineering (including approval of the ML access road intersection with the Burke Developmental Road); these permits would be completed during the detailed engineering phase ahead of commencement of construction, and are therefore also not considered to present constraints on the schedule. Other permits required during construction will include specialty trade permits and an operating permit for the plant.

The Owner will act as the general contractor and employ the services of subcontractors to perform the construction work in an open shop environment. This practice will ensure a competitive bidding process and allow the Owner to draw on a greater pool of contractors and construction workers.

Construction contracts will be written to protect this strategy and will include a non-affiliation clause to be signed off on by both the union and the unionized contractor.

The Project will be construction-driven, such that the scheduled site work will dictate the EP needs to meet the target completion date. The construction work force is estimated to peak at 460 workers and support personnel in the second year of construction. Most of this workforce will be skilled trades' persons.

There are existing camps in the area that can house construction workers in the very early stages of development. A self-supporting, 150-person temporary construction camp will be installed at the start of construction. This camp will house the initial onsite workforce, and will also be used to house the overflow from the operations village as required. The 300-person permanent operations village will be constructed at the site to support the entire operations team. During the construction period, the operations village, in conjunction with the temporary camp, will house the construction crews, Project management teams, and operations managers. The camp will provide single occupancy rooms with shared facilities for laundry, recreation, swimming pool, outdoor refreshment area, and dining halls.

24.1.2.6 Earthworks

As soon as detailed engineering begins, part of the early design will focus on site infrastructure. The early start of infrastructure engineering will support timely construction of the access road and interchange, camp access and site earthworks, transmission line, and surface water management structures. The construction strategy for earthworks is to use the mine production equipment to support the bulk earthworks program. The Feasibility Study capital cost estimate is based on using non-Potentially Acid Generating (non-PAG) construction material from the open pit.

The construction strategy, Project schedule, and capital cost estimate are based on mine equipment being used to construct the TSF embankment Zone C, and waste rock being supplied from the pit development for construction of uncomplicated earthworks constructions such as:

- Haul roads to the crusher, dumps, and TSF
- Mine and camp access roads
- Cabbage Tree Creek Diversion channel and berm
- Bulk fill behind the primary crusher (Mechanically Stabilized Earth [MSE] or Hilfiker-style wall)
- Supply oxide waste for the TSF basin liner and filter zones
- Supply competent waste rock for erosion protection.

The materials used for earth structures will come from the Non-PAG material available from the open pit. This strategy requires that a portion of the permanent mine fleet will need to be purchased (or leased) and made operational early in the construction program.

The capital cost advantage of using this strategy is that the cost per unit of material moved from the mine to the TSF represents only the difference in haul costs relative to those to haul mine waste to the waste dump.

Haul roads will be constructed to mine standards.

The plant site bulk earthworks are planned as a balanced cut-and-fill program. Site geotechnical investigations during the feasibility study have determined there are sufficient construction material types available on site to provide:

- Aggregate materials for general and structural backfill
- Sand for pipe and electrical cable bedding
- Filters for the tailings dam.

The mine will supply and operate an aggregate crushing and screening plant at the site to stockpile the various sizes and types of material needed for construction.

24.1.2.7 Concrete

Aggregate suitable for concrete is available in the area. The intent is to locate an 80 m^3/h concrete batching plant close to the new plant site, where most of the concrete will be needed. There will be several large pours of 12 to 16 hours' duration required, which include the crusher mat foundation, mill foundations, and mill piers. While the construction plan requires two or three concrete trucks with 5 m^3 to 8 m^3 capacity on a year-round basis, there will be a requirement to bring in additional trucks from local plants when the large pours are scheduled.

24.1.2.8 Steel and Buildings

Buildings will be minimized to providing only overhead protection. Generally, the process equipment will be open to the weather. Where substantial buildings are required, the Project will make use of pre-engineered buildings, or shelters.

24.1.2.9 Mechanical and Piping

Most of the capital equipment will be shipped via Townsville and Brisbane, particularly if sourced from offshore destinations. The PEP includes establishing a major freight forwarder to manage freight from the vendor's plant (domestic or international) to a laydown area on the site. Heavy loads include the transformers, crusher parts, mill, and motor parts. They will likely be break-bulk loads. The local bridges are capable of allowing these loads to pass over them safely, as long as they are high enough to clear the railings, and loaded onto multiple-axle low-bed trucks.

The bulk piping is to be purchased directly by the Owner's representative. Long-radius rubber-lined slurry lines and chutes will be pre-spooled and have the linings installed off site. The rest of the pipe will be spooled at site by the contractor, who will also prepare the isometrics. All manual valves will be purchased as bulk orders by the Owner, and all automatic valves, considered long delivery, will be purchased by the EP on behalf of the Owner.

24.1.2.10 Electrical and Instrumentation

All permanent secondary distribution, excluding overhead lines to remote areas from the main substation, will be installed by the contractor. This includes process and material handling electrical work. The electrical contractor will be responsible for the installation of motor control center (MCC) and switchgear. The management of Owner-approved lockout procedures is the responsibility of the electrical contractor until the handover to operations during the commissioning process.

A specialty line contractor will construct all overhead power lines.

The electrical contractor responsible for the secondary distribution will install the instrumentation and control system. The Owner's engineer will program the control system and commission it, in conjunction with the Owner.

24.1.3 Management Approach

Under the administration of the Owner's Project Manager, the Engineering (E), Procurement (P), and Construction Management (CM) team will manage the Project in accordance with the Project schedule, capital cost, health and safety, environmental, and quality targets.

The Owner will be responsible for safety, security, permits and licensing, mine planning and pre-production mining, communication and interaction with the local community and media, financing, accounting and invoice payment, operation staffing, operator training, wet commissioning, and start up.

The EP/CM team will provide detailed design, management of the construction program, reporting and Project cost control, scheduling of engineering and construction, and purchasing of all capital equipment on behalf of the Owner. Known equipment vendors will be requested to supply capital equipment in a competitive environment.

Contractors will be invited to bid competitively in an open shop environment such that the work will be open to unions, alternative unions, and non-union companies. This method has become the traditional method used in the construction of new mines, and affords the greatest potential workforce availability.

24.1.3.1 Staffing Plan

Management will be supplied by the Project team members as follows:

- Owner:
  - Project Manager
  - Project Engineering Discipline Leads
  - Document Control
  - Metallurgist
  - Mine Manager
  - Plant Superintendent
  - Environmental and Permit Manager
  - Controller
  - Human Resources Manager
  - Procurement Manager
  - Safety Manager
  - Site Security
- Engineering Procurement Consultant:
  - Engineering Manager
  - Procurement Manager
  - Metallurgist
  - Mine Planner

  - Engineering Leads
  - Project Engineering Controls
- Construction Management:
  - Construction Manager
  - Contract Administrator
  - Trade Managers
  - Field Engineering
  - Accounting/Cost Manager
  - Scheduler
  - Field Expediting/Procurement
  - Warehousing/Receiving
  - Camp Management
- Site Specific Consultants:
  - Site Surveying
  - Geotechnical for pit, major foundations, and tailings storage facility
  - Site Quality Assurance
  - Vendor Representatives: crusher, mills, lime, main motors, conveyors.

24.1.3.2 Project Procedures Manual

The Project development strategy must include the development of a comprehensive Project Procedures Manual encompassing the requirements and guidelines for the EP/CM work. The Project procedures manual will include the following:

- Project organization, key names, and communication procedures
- Identification of the division of responsibilities between the Project stakeholders
- Integration of the activities of other designers on the Project
- Reporting requirements, including Project systems, Project meetings and minutes
- Project data management (e.g., format, distribution, filing system)
- Engineering and drawing preparation and transmittal procedures
- Site procedures, including safety, environmental, and quality requirements
- Construction work item procedures
- Procurement and contract procedures
- Site and office personnel rules and regulations
- Cost reporting
- Scheduling
- Commissioning.

24.1.4 Constructability Reviews

Concurrent with the start of detailed engineering, construction planning and constructability reviews of engineering and procurement will begin. Drawings and documents will be reviewed to determine more effective construction methods and to establish parameters for a prefabrication or pre-assembly program. Once field work commences, construction will strive for continuous improvement based on activities such as team building, supervisory training, craft participation, problem solving, and iterative planning.

Engineering support during construction will be provided from the home offices of the Engineering Consultant under the general direction of the Owner's Engineering Manager. Support activities will include:

- Ensuring that packages of documents issued for construction are complete and up to date
- Ensuring that technical data and manuals are received and approved
- Resolving technical questions from the field
- Ensuring quality is achieved in the field and in fabrication shops
- Helping to expedite supplier information
- Checking that commissioning and normal and insurance spares are ordered and delivered
- Provide a resident engineer if required
- Collecting as-built records.

24.1.5 Construction Management

The Construction Manager manages all activities on the construction site to deliver the safe completion of the Project in accordance with the agreed scope of work, budget constraints, schedule, and defined quality and safety standards.

The Construction Manager provides team leadership and motivation, and manages the overall onsite construction effort by identifying priorities and setting goals, duties, and objectives. Regular meetings with the Owner, EP, and contractor site staff will be convened to clearly communicate the best construction outcome for the Owner.

24.1.5.1 Construction Management Objectives and Responsibilities

Key objectives for CM include:

- Employing site hazard management tools and programs to achieve zero accident / no harm Health Safety and Environment (HSE) objectives
- Applying contracting and construction infrastructure strategies to support the Project execution requirements
- Developing and implementing a construction-sensitive and cost-effective master Project schedule
- Establishing a field Project control system to ensure effective cost and schedule control, including a cost trending program
- Establishing a field contract administration system to effectively manage, control, and coordinate the work performed by the contractors

- Applying an effective field constructability program, as a continuation of the constructability reviews performed in the design office
- Developing a detailed field logistics and material control plan to maintain the necessary flow and control of material and equipment to support construction operations
- Meeting the schedule for handover of the plant to the commissioning team.

The CM team is responsible for:

- Field engineering
- Technical inspection of material and equipment delivered to site
- Contract supervision
- Surveys
- Managing site direct and indirect costs, cost system reporting, and Project controls
- Approving contractors' payment certificates
- Materials management and logistics
- Construction quality assurance and quality control (QA/QC)
- Inspection, commissioning, and system handover.

24.1.6 Contracts

The contracting approach will optimize the use of the local labour force, create a responsible and sustainable relationship within the community, and provide the right mix of management and specialist skills to support the safety, quality, and schedule and cost objectives of the Project. Contract packages will be designed to take advantage of a staggered work schedule, some of which may commence before full scope definition is complete; for example, construction of the power line, truck fuel station, and truck shop.

Owner-approved contract templates will be used for all equipment purchase orders, and construction and service contracts.

24.1.7 Cost Management

24.1.7.1 General

The Project budget will be based on the approved Feasibility Study capital cost estimate, scope of work, schedule, and quality plan, and will form the baseline against which progress and cost will be measured and managed.

The Project Manager will be responsible for cost management and reporting, and will provide an integrated Project management cost and schedule database for cost management and reporting.

24.1.7.2 Budget Allocation and Management

The budget for each work package is the estimated cost, scope, and time allocated from the Feasibility Study WBS. The aggregate of all work package scopes and allocated estimates forms the baseline for the measurement of performance, and the baseline for change control.

The Project will be managed by commitments, and the budget will be reconciled whenever significant cost commitments are made, including allowances for future variations and minor scope changes.

24.1.7.3 Contingency Management

Approval of transfers to and from the Project contingency account will be by the authority of the Owner. The account will include provisions for escalation and foreign exchange fluctuations, and will be managed on a global basis to fund approved cost increases within the Project scope. Contingency funds will not be used for Project scope changes.

24.1.7.4 Foreign Exchange

Most equipment and supplies will be purchased in Australia. Whenever feasible, imported items will be purchased from local agents, quoting Australian dollars, to minimize exposure to exchange rate risks.

At the time a definitive estimate is prepared for a package, changes in the relevant exchange rate will be compared to those in effect on the Feasibility Study estimate base date. In preparing the current budget for a work package, an element of the transfer to or from the contingency account may be used to compensate for exchange rate changes.

24.2 Project Schedule

24.2.1 General

The overall Project execution period from start (Project financing in place) to mechanical completion is approximately 22 months (Figure 24-1).

The Project schedule will be continually revised and updated. More detailed schedules will be developed for each work package, and will be used to revise the master baseline Project schedule. The detailed package schedules will consider interfaces, resource constraints, delivery times, contract scopes, detailed engineering and procurement times, and inputs from contractors. The resulting detailed Project schedule will be used to manage performance. Deviations from detailed schedules will be assessed monthly and used to measure impacts on the overall schedule.

Construction labour force is based on a 70-hour construction contractor work week, with crew rotations established by the contractors, which will generally be three weeks on site and one week off. Labour force loading indicates a peak requirement of 450 workers on site.

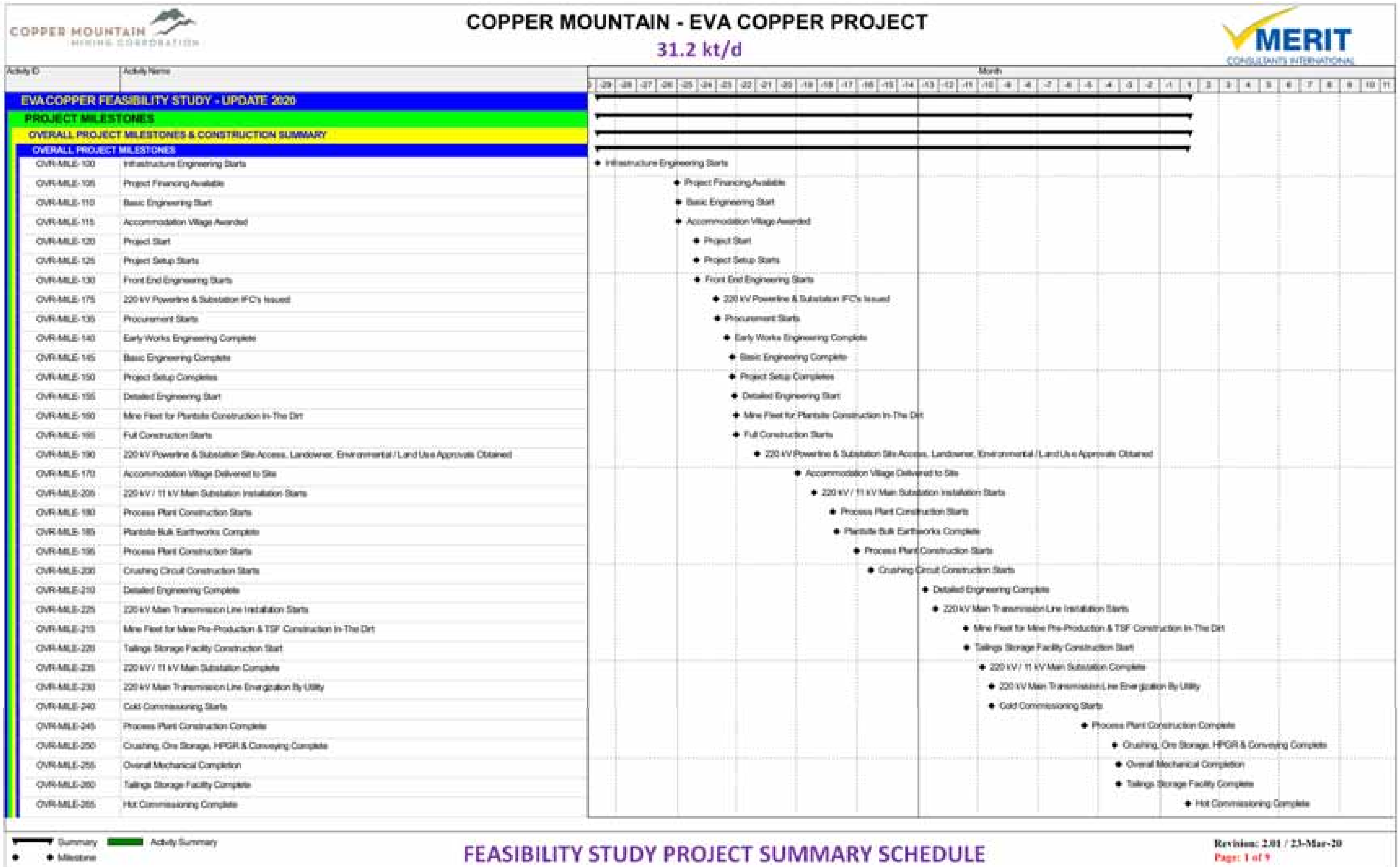


Figure 24-1: Project Schedule

24.2.2 Project Milestones

The Project schedule reflects a traditional approach to Project execution, with field construction commencing after engineering tasks are well advanced to accommodate long-lead times for the delivery of major equipment. Project milestones do not change once the Project is authorized. Progress is measured and reported compared to the fixed milestones, even if the schedule is adjusted. Notable Project milestones are listed in Table 24-1.

Table 24-1: Key Project Milestones

Milestone	Date
Early Infrastructure Engineering Starts	-29 months
Project Approval and Start	-25 months
Basic Engineering Complete	-22 months
Detail Engineering Complete	-12 months
Full Construction Starts	-22 months
Utility Power Required	-9 months
Tailings Storage Facility Complete	-3 months
Mechanical Completion	-3 months
Hot Commissioning Starts	-3 months
Commercial Production Starts	Month 1

24.2.2.1 Construction Permitting

The Owner’s team is responsible for all regulatory components of Project approval, and will be the primary liaison with all regulatory bodies for permanent plant and equipment permitting. The EP/CM will provide engineering support to the Owner to assist with permitting, and the Owner will develop a schedule and communicate status regularly to the EP/CM.

Individual contractors will be responsible for providing permits required for their construction activities.

24.3 Quality Control and Quality Assurance

24.3.1 Manufactured Items

Equipment suppliers will be required to provide details of their QC systems at the time of bidding. Quality systems will be considered in the bid comparison and vendor selection process. The level of detail and the level of involvement of the Project team will depend on the complexity and cost of the equipment involved.

24.3.2 Construction

Contractors will be required to nominate their QC plans at the time of bidding. The Engineering Manager, who has overall responsibility for quality, will review contractors’ QC plans and direct revisions where necessary. Approved QC plans will become part of contract documents.

24.3.3 Risk Management

The EP/CM Project Manager will ensure that the list of possible risks is reviewed and that the agreed risk reduction plans are implemented through detailed design, construction, and commissioning. Some risks are operational risks, and are the responsibility of the Owner's operational team. All risks will be entered in the Risk Register, together with agreed mitigation plans and the person responsible. Whenever possible, risks will be addressed through design.

24.3.4 Health, Safety, Environment

24.3.4.1 Safety Management Plan

The Project Manager, via the Site Safety Manager, is responsible for ensuring that a Safety Management Plan is in place before construction begins. The plan will meet the requirements of applicable safety, health, environmental, hygiene, and emergency response legislation.

Key features of the plan are:

- Contractor plans must at minimum comply with the requirements set out in the Project safety plan.
- Contractors will be held accountable for enforcing their plans, including discipline.
- Workers will be primarily responsible for their own safety, and will be provided information and education to ensure such.
- Site-specific induction will be given before the first working shift, i.e., upon arrival.
- Zero tolerance will be given for infractions of policy.
- Contractors will be responsible for providing properly trained personnel to perform the work. The Safety Manager will inspect proof of training, licenses, and qualifications, and maintain records. These will be submitted during the contract negotiation stage, and before a contractor is permitted to mobilize.
- The EP/CM Project manager or Safety Manager will have the right to eject from the site for cause any contractor's employee and to demand a replacement at the contractor's expense.

24.3.4.2 Safety Inductions

All personnel arriving on site will attend site induction upon arrival. Induction will include an overview of the Project, site rules, and emphasize the Projects' commitment to safety and environmental protection.

24.3.4.3 Health and Hygiene

The Project will provide a first aid facility, showers, and washing and toilet facilities to meet regulations. Temporary sanitary facilities (chemical toilets) will be provided close to work places until the permanent facilities are operational.

24.3.4.4 Emergency Preparedness and Response

Trained and certified personnel will provide first aid for the Project. First aid facilities will be provided at the temporary construction office until the permanent facility in the service complex is available.

Ambulance service will be available from the mine site. A mine rescue room will be established, and rescue equipment purchased.

An emergency response plan for environmental spills will be established upon Project start according to established environmental practices.

24.3.4.5 Environmental Management

The Owner is responsible for providing a comprehensive Management Plan for items such as:

- Air quality
- Water quality
- Waste handing and disposal
- Waste rock and tailings storage
- Ecosystems and vegetation protection
- Wildlife protection
- Aquatic resources protection
- Metal Leaching and Acid Rock Drainage (ML-ARD) prevention and mitigation
- Surface subsidence
- Dust control
- Archaeology
- Noise control.

The EP/CM will ensure that these plans are enforced, periodically reviewed, and updated by the Project team. Project organization includes an environmental coordinator and a technician (both part time) reporting to the Manager of Safety and Environment.

24.3.5 Temporary and Site Facilities

There will be a number of temporary components used for the construction and pre-preproduction periods that will be removed once plant operations start and the construction team leaves the site. Some temporary works will be incorporated into the permanent facility and not be removed as the plant goes into operations. Overall, the Project development commitment is to return the environment, not disturbed by the permanent plant, back to a condition that is environmentally acceptable.

Once construction begins, there will be a need to establish site offices and communications for the Project development team to set up close to the work.

Support services will be established for power, water, and sewage. Diesel generators will be used for construction.

The site construction offices and laydown areas will be located within walking distance of the process plant. Workers will be taken by bus from the accommodation village (camps) to the work locations.

Construction fuel tanks will be installed in suitable lined containments at site. Fuel will be delivered as required from the nearby community, and a site fuel bowser will fuel remote day tanks.

The transportation, storage, dispensing, and use of fuels at the site will be conducted in compliance with all relevant government laws and regulations.

24.3.6 Security

Overall security of the Project construction at site will be the responsibility of the Owner. A security service contractor will be retained, and an entry gate will be built to ensure the physical security of the facilities, and to control and record the access of people to construction and restricted areas. Vehicles and people will be inspected on the way in for fitness to work and authority to enter, and on the way out to check for unauthorized removal of materials from the construction site.

24.3.7 Commissioning

For this Project, commissioning applies to fixed plant and equipment, but not to mobile mining equipment. Commissioning of specific systems will be carried out by coordinated teams formed for that purpose from:

- Project Construction Management
- Owner's Operations and Maintenance team
- Project design engineering
- Suppliers
- Construction contractors.

A commissioning program will be developed by the Commissioning Manager, and will contain key commissioning definitions, an outline of the facilities to be commissioned, a summary plan, guidelines on risk management and QA/QC, and samples of a number of sign-off certificates. The Commissioning Manager will ensure that individual system commissioning plans are started when design engineering is about 75% complete. A standard plan template will be used for each system plan.

The Commissioning Manager will establish and maintain the overall commissioning program, support development of system commissioning plans by engineers, provide support for commissioning, and help ensure the consistent application of the program.

The main responsibilities of a system commissioning team (several teams, for different systems at different times) are:

- Prepare the commissioning plan for the specific system
- Track the resolution of items on the defect list, including scope defects
- Organize supplier commissioning assistance from vendor representatives
- Participate in inspections, testing, and trials
- Participate in safety checks
- Prepare turn-over packages to client of commissioned systems
- Sign certificates of acceptance as systems are released from the contractor for cold-commissioning dry runs
- Coordinate the transition to operations personnel during hot commissioning and ramp-up.

24.3.7.1 Pre-Commissioning: Commissioning of Equipment

This stage of commissioning consists of the complete inspection, testing, and operation of each piece of equipment individually, checking that electrical control and power wiring has been connected to the equipment correctly, and checking the configuration and calibration of each instrument loop. The constructor is responsible for directing this stage. The construction Trade Managers will witness tests, approve inspections, and countersign the checklists and data sheets.

24.3.7.2 Cold Commissioning: Commissioning of Systems

This stage consists of testing and operating the equipment, grouped together into systems or facilities without product, e.g., the water system without water, or crushing and screening without rock. The contractor is responsible for directing this stage. At the end of the stage, the contractor will have corrected all defects identified by the construction Trade Manager deemed necessary to proceed with hot commissioning. Upon completion of cold commissioning, the contractor, Owner’s Operations and Maintenance senior representative, and the Project Construction Supervisor will sign a certificate accompanied by a defect list that hands the system over to the CM team.

24.3.7.3 Hot Commissioning: Start-Up

In this stage, Operations personnel will put the facility into operation with mined materials, reagents, and fluids, under the direction of the Project team. The objective is to prove that the system will operate acceptably under realistic conditions. Assistance from the contractor at this stage will be considered extra work, and will be covered under a separate contract.

24.3.7.4 Acceptance and Production Ramp-Up

Ramp-up of the entire mine and mill complex will be achieved under the direction of the Operations management team once the final acceptance certificates for individual systems have been signed off and ownership is transferred. Operations will assume responsibility for detailed planning and execution of the ramp-up, together with correction of any remaining defects. Detailed planning for ramp-up will begin before commissioning is underway.

24.3.8 Project Completion

The Project close-out plan will include:

- Transfer of care, custody, and control
- Issuing Project close-out notices to all vendors
- Declaring Project personnel redundant with appropriate notice periods, according to an agreed schedule
- Obtaining waivers and lien releases
- Processing and resolving all remaining change orders, claims, disputes, back charges, and final payments
- Releasing holdbacks
- Confirming that the Owner has a complete set of:
  - Warranty records

  - QA records
  - As-built record drawings
  - Operations and maintenance manuals
  - Parts lists
  - Compliance documents (e.g., permits, licenses, and inspection reports)
- Closing the Project accounting system, and cancelling the delegation of authority
- Issuing the Project close-out report, including the final cost report.

All Project completion activities are expected to be substantially completed when the Project meets its production objectives.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology, Mineral Resources, and Mineral Reserves

- The Eva Copper Project Mineral Resources are IOCG deposits that vary according to setting. The main deposit, Little Eva, is similar to Ernest Henry.
- Mineralization primarily occurs as chalcopyrite.
- The mineralized zones typically trend north to south, and are moderately to steeply dipping.
- The Mineral Reserves listed in Table 15-1 comply with all disclosure requirements for Mineral Resources set out in NI 43-101.
- Copper Mountain Mining Corp. (CMMC) and Stuart Collins, P.E., believe the Mineral Reserves are being estimated in an appropriate manner using current mining software and procedures consistent with reasonable practices. The classification of Measured, Indicated, and Inferred Resources conforms to Canadian Institute of Mining, Metallurgy, and Petroleum Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM Definition Standards).
- Mr. Collins is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that would materially affect the Mineral Reserve estimates.

25.2 Mining

- Conventional open pit mining methods (drilling, blasting, loading, hauling) will be employed to extract the ore and waste.
- There are seven deposits to be mined: Little Eva, Turkey Creek, Bedford, Lady Clayre, Ivy Ann, Blackard, and Scanlan. None of the deposits has previously been mined. Little Eva, Turkey Creek, Bedford, Lady Clayre, and Ivy Ann represent approximately 70% of the Mineral Reserves.
- Mining by CMMC personnel will begin in the Little Eva pit (Year 1), and in the Blackard and Turkey Creek pits from Year 3 onwards. Mine life is 15 years, with a one-year mining preproduction period. The Project's overall strip ratio (waste tonnes to ore tonnes) is 2.2:1.
- The mine plan estimates that there are 170 Mt of ore grading 0.46% Cu and 0.05 g/t Au, and 381 Mt of waste will be generated over the LOM.
- Topographical relief, climate, haul distances, and geographic location present no issues to the Project.
- Factors that could impact production, if not addressed by CMMC, are dewatering the pit and slope stability.

25.3 Metallurgical Testwork and Mineral Processing

- The competency and hardness values for the 75:25 blend of sulphides and native copper ore sources indicates 31,200 t/d at 165 µm grind is achievable with the updated plant design.
- Little Eva, being the largest source of sulphide ore, is expected to see 95% Cu recovery. The remaining sulphide ore sources are expected to see between 88% to 95% recoveries, depending on the mineralogy.

- Blackard and Scanlan native copper zones are expected to achieve 63% recovery through gravity and flotation recovery methods
- The recovery within the native copper zone of Blackard will be variable; however, it will average 63%, as shown in the testwork. The sulphide zone located below this, is expected to behave similarly to Turkey Creek, at an anticipated 88% recovery.
- Extensive work has been done on Blackard. Scanlan has not seen the same degree of study; however, pilot flotation work and geological observations on Scanlan have shown it to have similar mineralogical characteristics as Blackard.

25.4 Process Plant

- The process plant flowsheet is a standard processing plant design, featuring a format of two crushers and HPGR with a gravity recovery circuit installed on the ball mill and regrind cyclone loops.
- The processing plant has been designed to produce a marketable copper concentrate grade of 28% Cu and about 3 g/t Au.
- The daily average throughput is 31,200 t/d over the mine life based on geometallurgical projections of the mine plan.

25.5 Infrastructure

This greenfield project will require the following major components to be built:

- Access and site roads
- Accommodation village for Project construction and operational personnel
- An 11.44 Mt/a capacity crushing, milling, and flotation process plant
- An 11 km, 220 kVA transmission line, substation, and site distribution electrical system
- A water supply system to provide approximately 19,000 m^3/d of water
- Site administration office complex, and a six-bay truck and plant maintenance shop with attached warehouse facilities
- Tailings storage facility (TSF)
- Site sediment management installations
- Cabbage Tree Creek diversion channel around the Little Eva pit, and surface water bunding
- Fuel storage and dispensing
- Plant site laboratory
- Communications facilities
- Training and first aid facilities
- Open pit mining infrastructure
- Borefield dewatering wells for the open pits, and the Cabbage Tree Creek supply
- Explosives bulk storage depot and magazine.

25.6 Environmental, Permitting, and Social Considerations

- Mining Leases (ML) and an Environmental Authority (EA) for the Project have been granted. The EA from the Department of Environment and Science (DES) regulates the environmental management of the Project and sets out key environmental management conditions. The current EA is based on the previous 2016 mine layout. Changes to the mine layout and throughput increases set out by this Feasibility Study update will require submission of a Major Amendment, and have the potential to trigger an Environmental Impact Statement (EIS) review which are expected in a timely manner. These are straightforward procedural processes.
- To support EA applications, all baseline studies (like flora and fauna surveys, or waste and tailings rock characterization) have been undertaken, and these included work to support mining of the open pits, and location of the waste dump, TSF, mine access road, and Cabbage Tree Creek diversion bund and channel.
- The Project area is uninhabited, with the closest sensitive receptor being Mount Roseby Homestead, which is approximately 17.5 km southeast of the Little Eva pit and processing plant, and 1.1 km from the Scanlan pit. Noise and air quality monitoring is a requirement of the EA.
- The key risks associated with release of contaminants into the environment have been considered, with the design incorporating surface water management control dams and inclusion in the TSF design of a low-permeability basin, cut-off drains, and monitoring.

25.7 Capital and Operating Costs

- Approximately 350 full-time jobs will be directly created by this Project.
- Initial capital costs are approximately $454.5 million, and sustaining capital costs are estimated to be $34.0 million at an assumed exchange rate of AU$1.55 to US$1.
- Average LOM operating costs are estimated to be $11.39/t milled (excluding royalties). The C1 cash cost is estimated at $1.44/lb.

25.8 Economics

- The Project has a recoverable copper content of 1,502 Mlb of copper and 205 koz of gold over a 15-year life.
- Project economics are good at a long-term copper price of $3.04/lb and a long-term gold price of $1,362/oz.
- A long-term exchange rate of AUS$1.55 to US$1 was used.
- At a discount rate of 8%, the after-tax NPV is $437 million, and the after-tax IRR is 29%.
- This Project is most sensitive to the copper price, copper recoveries, and copper head grade delivered to the process plant. The exchange rate, operating costs, and capital costs may also impact the Project's economics to a lesser degree.

26 RECOMMENDATIONS

26.1 Mineral Resources and Mineral Reserves

- Drill targets below and within the current pit designs to convert Inferred Resources to Indicated Resources.
- At the Little Eva pit, conduct development drilling ahead of mining to improve the quality of the Mineral Reserves, and optimize mining selectivity and grade control costs/strategy.
- Perform geotechnical slope studies on the Blackard, Scanlan, Turkey Creek, Lady Clayre, Bedford, and Ivy Ann deposits.
- Continue detailed mine design and mine planning on the Eva Copper Project prior to production.
- Develop detailed dewatering plans for the Little Eva, Blackard, Scanlan, and Turkey Creek pits.

26.2 Infrastructure, Process, and Plant

- Perform confirmatory geotechnical investigation of Cabbage Tree Creek bund and the tailings storage facility (TSF) second cell western side.
- Re-evaluate the hydrology and dewatering of the Little Eva, Blackard, and Turkey Creek pits in the context of the new geotechnical models.
- Redo the overall site Hydrogeology Report, last done by KH Morgan in December 2009, to include the Cabbage Creek borefield and potential water bore source for the accommodation village.
- Perform follow up testwork to investigate further improvement of final grade by means of magnetic separation. Some testwork has highlighted that this is an effective means of removing iron bearing minerals, barren of copper, from final concentrate during coarse gravity separation. This combined with additional investigation into the cleaner circuit could yield further improvement on final product grades, improving the economics of the Project.
- Investigate the potential of the gravity concentrate bypassing the smelting process, which might attract a slightly elevated price per tonne.
- Scanlan ore was studied during bench and pilot tests performed in 2006. There is no recent data on this ore source; however, all data and geological observations indicate equivalent behaviour to Blackard ore. Additional testwork and spatial variability investigations should be performed to enhance the understanding of this deposit, despite the mining plan indicating mining of Scanlan ore will only start in Year 7. There is no data available on the deeper sulphide portion of this deposit.

26.3 Project Environmental Authority (EA EPM L00899613)

Both the Mining Leases (ML) and the Environmental Authority (EA) have been approved. Changes made to the mine layout in this Feasibility Study require a new amendment to the existing EA. Amendments are assessed to determine whether they are classified as Minor or Major. The extent of the new mine footprint, increased processing throughputs, adjustments to the waste dump, plant areas, TSF, Cabbage Tree Creek water well field, and road routes, and inclusion of the Blackard and Scanlan deposits to the mine plan will require submission of a Major Amendment Application to the existing EA. From the date of application submission, the Minor Amendment process takes up to 35 days, while the time for a Major Amendment can vary. The 2016 Major Amendment by Altona took 3.5 months from the date of application submission.

27 REFERENCES, ACRONYMS, AND UNITS OF MEASURE

27.1 References

AARC (2007). Roseby Copper Project, Environmental Impact Statement (EIS) Supplementary Report. (L2800).

AARC (2008). Roseby Copper Project, Environmental Impact Statement (EIS) Response to Information Request. (L2800).

AARC (2011). Roseby Copper Project, Environmental Management Plan. (L3428).

AARC (2019). Eva Project, Ecology Report. (L10479)

ALS AMMTEC (2012a). Metallurgical Testwork conducted upon Copper Ore Samples from the Roseby Copper Project for Altona Mining Limited Report A13828 September 2012. Perth: ALS AMMTEC. (L8464).

ALS AMMTEC (2012b). Quantitative Automated Mineralogical Analysis conducted on Five (5) Copper Core Samples from the Roseby Project for Altona Resources (Project A14159). Perth: ALS AMMTEC. (L7842).

ALS AMMTEC (2012c). Quantitative Automated Mineralogical Analysis conducted on Two (2) Flotation Feed and Two (2) Flotation Tail Samples for Altona Resources (Project A13828). Perth: ALS AMMTEC. (L7844).

ALS Metallurgy (2016). Metallurgical Testwork conducted upon Turkey Creek and Little Eva Copper Ores. Perth: ALS Metallurgy. (L9654).

Altona & Department of Natural Resources and Mines. (2017). Query to the Department of Natural Resources and Mines, Queensland, about Water License requirements for the Little Eva Project. Queensland: Altona & DNRM) (L9714).

Altona Mining Limited (Altona) (2014a). ALT2014004/005 Turkey Creek Drill Programme Completion Report. Perth: Altona (L9719).

Altona (2014). Little Eva Mineral Resource Update. Perth: Altona (L9111).

Altona (2014a). Little Eva Geotechnical Review Version 2. Perth: Altona. (L5561).

Altona (2014b). Little Eva Geological Report. Perth: Altona (L9110).

Altona (2014b). Review of Little Eva Unconfined Compressive Strength Measurements. Perth: Altona. (L9759).

Altona (2017). ALT2015009 Metallurgical Drilling Programme Completion Report. Perth: Altona (L9748).

Altona (2017). Detailed Metallurgical Review, Cloncurry Copper Project. Perth: Altona. (L9782).

Altona (2017). Detailed Metallurgical Review. Perth: Altona. (L978).

Altona (2017). Query to the Department of Natural Resources and Mines, Queensland, about Water License Requirements for the Little Eva Project. Perth: Altona. (L9714).

AMML (2007a). Bond Ball Mill Work Index Tests on Little Eva Ore Samples for Universal Resources Limited. Perth: AMML. (L2902).

AMML (2007b). AMML Interim Report 0013-2, Batch and Locked Cycle Flotation on Little Eva Ore Samples. Sydney: AMML. (L2901).

AMMTEC (2006a). Comminution Testing Conducted Upon Samples from the Roseby Copper Project for Universal Resources Limited, Report A10110. Perth: AMMTEC. (L1356).

AMMTEC (2007b). Comminution Testwork Conducted Upon Sulphide Composite Samples from the Little Eva Copper Deposit (Roseby Copper Project) for Universal Resources Limited, Report A10997. Perth: AMMTEC. (L2904).

AMMTEC Consultants PLLC (AMMTEC). (2005a). Metallurgical Testwork Conducted Upon Samples of Ore from the Roseby Copper Deposit, Report A9761. Perth: AMMTEC. (L1260).

AMMTEC. (2005b). Flotation Testwork on Bedford, Little Eva and Lady Clayre Samples for Universal Resources, Report A9656. Perth: AMMTEC. (L3380).

AMMTEC. (2006b). Bench Scale Flotation Testwork on Roseby Ore Samples for Universal Resources Limited, Report No. A9880, Part B. Perth: AMMTEC. (L1352).

AMMTEC. (2006c). Pilot Grinding and Preliminary Pilot Flotation on Ore Sample Composites from the Roseby Copper Project for Universal Resources Limited, Report No. A9880 Part C. Perth: AMMTEC Ltd. (L1353).

AMMTEC. (2007). Flotation Testwork on Blend Master Composites for Universal Resources Limited. Report No. A10851-D, Perth: AMMTEC. (L2905).

Ausenco (2020). Process Plant and Facilities Layout Drawings, Ausenco, February 2020.

AustralAsian Resource Consultants (AARC). (2007). Roseby Copper Project, Environmental Impact Statement (EIS). (L2800).

Browning, F. (2016). Thesis: Geological Model and Resource Estimate for the Bedford Copper-Gold Deposit, Mount Isa Inlier, North West Queensland. Cornwall: University of Exeter (L9754) [Report Only].

Chadwick, R. (1992). Ivy Ann Prospect, EPM 8059 "Cameron River," 1992 Exploration Summary and Discussion. Perth: Dominion Mining Limited. (L987).

CITIC-SMCC Process Technology (2018). Review of the Equipment Selection for the Comminution Circuit of the Cloncurry Project, January 2018.

Copper Mountain Mining Corporation (CCMC). (2018). Copper Mountain Concentrator, July 2018. Eva Flotation Testwork Progress Report No. 1.

CRAE. (1996). The Geology and Origin of the Blackard, Lady Clayre and CRA Flat Cu and Cu-Au Prospects within the Mount Roseby Area. Brisbane: CRA Exploration Pty. Ltd. (L852).

Department of Environment and Heritage Protection (DEHP), Queensland. (2012). Environmental Authority MIN102973311 (ML90162, ML90163, ML90164, ML 90165, and ML90166) for Altona Mining Limited and Roseby Copper Pty. Ltd. Cairns: DEHP. (L8299).

DEHP, Queensland. (2013). Amendment of Environmental Authority EPML00899613 (ML90162, ML90163, ML90164, ML 90165, and ML90166) for Altona Mining Limited and Roseby Copper Pty. Ltd. Cairns: DEHP. (L8299).

DEHP, Queensland. (2016). Amendment of Environmental Authority EPML00899613 (ML90162, ML90163, ML90164, ML 90165, and ML90166) for Altona Mining Limited and Roseby Copper Pty. Ltd. Cairns: DEHP. (L8299).

EHW (1996). Report on Field Mapping and Proposed Diamond Drill Holes at Ivy Ann. Perth: EHW. (L991).

Environmental Geochemistry International. (2006). Geochemical Characterisation and ARD Assessment of Samples from the Roseby Copper Project. Perth: Environmental Geochemistry International. (L2959).

Gekko Systems (Gekko) (2019). Eva Copper Project Gravity and Flotation Testwork, December 2019

Geological Survey of Queensland. (2011). Northwest Queensland Mineral and Energy Province Report. Brisbane: Department of Employment, Economic Development and Innovation (L8418) [Report Only].

George Orr and Associates (George Orr) (2006). Little Eva Deposit: Geotechnical Evaluation for Mining Feasibility Purposes, July 2006, updated 2012.

George Orr (2006). Roseby Copper Project: Little Eva Deposit, Geotechnical Evaluation for Mining Feasibility Purpose. Perth: George Orr. (L1349).

George Orr (2012). Roseby Copper Project, Little Eva Deposit, Definitive Mining Feasibility Study: Geotechnical Evaluation for Open Pit Mining. Perth: George Orr. (L7821).

Gilbride Management (2011). Altona Mining Limited, Little Eva Project, Definitive Feasibility Study, Appendix 9.1, Logistics Study. Perth: Gilbride Management. (L8443).

GR Engineering Services (GRES) (2012). Altona Mining Limited, Little Eva Project, Definitive Feasibility Study. Perth: GRES. (L8443).

GRES (2015). Turkey Creek Slighter Testwork. Perth: GRES. (L9347).

Habermann, P. (1999). Thesis: Alteration and Mineralization at the Lady Clayre Cu-Au Prospect, Mount Isa, Eastern Succession, NW Queensland. Townsville: James Cook University. (L1064).

Hatch (2018). The Eva Copper Project Feasibility Study Report, Copper Mountain Mining Corp, November 2018.

Hatch (2018). 3D Process Plant Layout Drawings for the Eva Copper Feasibility Study. Vancouver: Hatch.

Hatch (2018). Electrical Single Line Diagrams for the Eva Copper Feasibility Study. Vancouver: Hatch.

Hatch (2018). Mass Balance and Water Balance for the Eva Copper Feasibility Study. Vancouver: Hatch.

Hatch (2018). Mechanical Equipment List for the Eva Copper Feasibility Study. Vancouver: Hatch.

Hatch (2018). Process Design Criteria for the Eva Copper Feasibility Study. Vancouver: Hatch.

Hatch (2018). Process Flow Diagrams for the Eva Copper Feasibility Study. Vancouver: Hatch.

Kendrick, S. (2013). Email: Altona's Little Eva Project Preliminary Questions [GRES-PROJECTS.FID2795]. Perth: GRES. (L8907).

KH Morgan and Associates (KH Morgan) (2009). Hydrogeology Report, December 2009.

KH Morgan and Associates (KH Morgan) (2011). Hydrogeology Report, Environmental Management Plan, Roseby Copper Project, Northwest Queensland. Perth: KH Morgan. (L1536).

KH Morgan (2012). Dewatering Procedures Impact and Water Resources, Little Eva Pit, Altona Mining Limited. Perth: KH Morgan. (L8414).

Klohn Crippen Berger (KCB) (2019). Eva Project Tailings Characterization Factual Report, September 2019.

KCB (2010). Definitive Feasibility Study review and update of the Tailings Storage Facility for the Eva Copper Project, Klohn Crippen Berger, December 2019.

Knight Piésold Pty. Ltd. (Knight Piésold) (2012). Altona Mining Limited, Little Eva Project, Definitive Feasibility Study, Appendix. Perth: Knight Piésold.

Knight Piésold (2016). Tailings Storage Facility Preliminary Design Report (Incorporates Regional Surface Water Management Design). Perth: Knight Piésold. (L9566).

Knight Piésold (2016a). Tailings Storage Facility Preliminary Design Report (Incorporates Regional Surface Water Management Design). Perth: Knight Piésold. (L9566).

Knight Piésold (2016b). Cabbage Tree Creek Inundation Study and Little Eva Pit Diversion. Perth: Knight Piésold. (L9567).

Knight Piésold (2018). Tailings Storage Facility Definitive Feasibility Study, Book A. Geotechnical, Tailings, and Water Management Report; and Book B. Geotechnical Interpretative Report, October 2018.

Knight Piésold (2019). Eva Copper Project, Cabbage Tree Creek Inundation Study and Little Eva Pit Diversion. Perth: Knight Piésold. (L10498).

Knight Piésold (2019). Eva Project, Definitive Feasibility Study – Geotechnical, Tailings and Water Management Report. Perth: Knight Piésold. (L10397).

Knight Piésold (2019). Eva Project, Definitive Feasibility Study – Geotechnical Interpretative Report. Perth: Knight Piésold. (L10399).

Knight Piésold (2019). Cabbage Tree Creek Inundation Study and Little Eva Pit Diversion Update, Knight Piésold, December 2019.

Lane, G., Foggiatto, B. and Bueno, M.P. (2013). Power-based comminution calculations using Ausgrind. In Alvarez, M., Doll, A., Kracht, W. and Kuyvenhoven, R. (Eds.), Proceedings of 10th International Mineral Processing Conference (Procemin 2013), pp. 85-96. Santiago: Gecamin.

LAROX (2008). Test Report 7.2.12 – Pilot Produced Final Copper Concentrate. Perth: LAROX. (L3294).

Li, C. (1996). Lady Clayre Prospect – Locked Cycle Testing. Melbourne: CRA ATD. (L930).

Mason Geoscience (2003). Petrographic Descriptions for Twenty-two Drill Core Rock Samples from Lady Clayre and Little Eva Prospects (Eastern Succession, Mount Isa Inlier, Queensland) Report 2811. Adelaide: Mason Geoscience. (L1058).

Mason Geoscience (2004). Petrographic and Minerographic Descriptions for Sixteen Drill Chip Rock Samples from the Little Eva, Lady Clayre, Bedford and Tin Lizzie Prospects (Roseby Project, Cloncurry Region, Queensland). Report # 2966. Adelaide: Mason Geoscience. (L1061).

Mason Geoscience (2005). Mineralogy of Three Cu-Bearing Ores: LED197, LED198, and BC Master Composite (Roseby Project, Queensland). Perth: Mason Geoscience Pty. Ltd. (L1188).

MBS Environmental (MBS). (2011). Roseby Copper Project Tailings Characterization and Acid Mine Drainage Management. Perth: Martinick Bosch Sell Pty. Ltd. (L3402).

MBS (2011). Roseby Copper Project, Waste Rock Characterisation and Acid Mine Drainage Management. Perth: MBS. (L3403).

MBS (2012). Altona Mining Limited, Little Eva Project, Definitive Feasibility Study, Appendix 10.2, Geochemical Assessment of Little Eva Tailings. Perth: MBS. (L8443).

MBS (2012). Roseby Copper Project, Surface Water and Sediment Quality Monitoring (March 2012). Perth: MBS. (L8613).

MBS (2015). Roseby Copper Project 2014/15 Water and Sediment Quality Monitoring Summary. Perth: MBS. (L9352).

MBS (2016). Little Eva Project, Environmental Authority Amendment Application Supporting Document. Perth: MBS. (L9609).

MBS (2016). Little Eva Project, Environmental Authority Amendment Application Supporting Document. Perth: MBS. (L9609).

MBS (2016). Memorandum: Environmental Approvals, Little Eva Project. Perth: MBS (L9768).

MBS (2016). Memorandum: Environmental Offsets Act 2014. (L9767).

MBS (2016). Roseby Copper Project 2015/16 Water and Sediment Quality Monitoring Summary May 2016. (L9655).

MBS (2016). Waste Rock Characterisation, Little Eva Project, Satellite Deposits (Turkey Creek, Bedford, Lady Clayre). Perth: MBS. (L9569).

MBS (2017). Cloncurry Copper Project 2016/17 Water and Sediment Quality Monitoring Summary. (L9772).

Merit and Sedgman (2017). CMMC Business Case Analysis Completed in October 2017.

Metso Corporation (Metso) (2019). Eva Copper HPGR Testwork (April 2019), and Little Eva-Blackard Blend Testwork, November 2019.

Milligan, S. & C. Li. (1996). Lady Clayre Prospect – Metallurgical Characterisation, Melbourne: CRA ATD. (L0842).

MMG Limited (2019). Mineral Resources and Ore Reserves. Available at: https://www.mmg.com/our-business/mineral-resources-and-ore-reserves/, accessed April 23, 2020.

NeoProTec Pty. Ltd. (NeoProTec) (2003). Testwork Report for Universal Resources Limited. Metallurgical Testwork Program, Little Eva, UR0301. Perth: NeoProTec. (L3022).

NeoProTec (2003b). Testwork Report for Universal Resources Limited. Metallurgical Testwork Program, Little Eva, UR0301. Perth: NeoProTec. (L3440).

NeoProTec (2005). Preliminary Flotation Response of: 1. Little Eva Low-grade and its Revenue Impact on Little Eva. 2. Lady Clayre. 3. Bedford. Ref: UR 05011, Perth: NeoProTec Pty. Ltd. (L1175).

NeoProTec (2006). Roseby Project. Comminution Testwork. Perth: NeoProTec. (L1388).

NeoProTec (2008). Roseby Copper Project – DFS Metallurgical Summary Report. Perth: NeoProTec.

Optimet Laboratories (Optimec) (2003). Metallurgical Testing of Roseby Copper Project Report P0013. Adelaide: Optimet. (L1580 & L1581).

Optimet (2005). Report P0117 Universal Resources Little Eva/Roseby Project Oxide/Sulphide Blend Flotation. Adelaide: Optimet. (L1577).

Orway Mineral Consultants. (2006). Universal Resource Ltd. Roseby Copper Project Feasibility Study Comminution Circuit Design, Report 46203. Perth: Lycopodcium Engineering Pty. Ltd. (L8434).

Outotec (2008). SUPAFLO® Thickener Test Data Report S394TA. Perth: Outotec. (L3292)

Paterson & Cooke (2019). Blackard Ore Pumping Conceptual Study, November 2019.

Paterson & Cooke (2019). Eva Tailings Dewatering Conceptual Study, November 2019.

Paterson & Cooke (2019). Tailings and Ore Characterization, November 2019.

Queensland Department of Environment and Heritage Protection (2016). Amendment of Environmental Authority EPML00899613, July 2016.

Rockwater Hydrogeological and Environmental Consultants (Rockwater). (2018). Water Supply Investigation for the Eva Copper Project.

Rockwater (2011). Little Eva Copper-Gold Deposit Water Supply Investigation. Results or RC Exploration Drilling. Perth: Rockwater. (L7816).

RPS Group plc. (2019). Little Eva Mine Site LiDAR. RPS Reference PR139400-1. (L10479).

SRK Consulting (SRK) (2014). Review of the Roseby Project Queensland. Perth: SRK (L9096).

Sedgman Limited (Sedgman) (2017). Cloncurry Copper Project Business Case Analysis, CMMC, and Merit, October 2017.

Taylor, R.G. (2009). Petrological Observations and Overview Comments Concerning 18 Samples from the Little Eva Copper-Gold Prospect, Cloncurry Region, Queensland, Australia. Townsville: School of Earth and Environmental Science, James Cook University. (L8850).

Townend (2015). Semiquantitative Analysis of Eight Metallurgical Samples. Perth: Townend Mineralogy Laboratory. (In L9347).

Universal Resources Inc., Como Engineers Pty. Ltd., & GR Engineering Services (2009). Roseby Copper Project 5MT/A Definitive Feasibility Study. (L1585).

Walker, Newman & Associates (2012). Altona Mining Limited, Little Eva Project, Definitive Feasibility Study, Appendix 9.2, Telecommunications and IT Cost Estimate. Perth: Walker, Newman & Associates. (L8443).

Withnal, I.W. & L.C. Cranfield (2013). Geological Framework of Queensland. Geological Survey of Queensland. Brisbane: Department of Natural Resources and Minerals.

Woodgrove Technologies (2019). Eva Project DFR Pilot Plant Report, May 2019.

27.2 List of Acronyms and Units of Measure

27.2.1 Acronyms

Acronyms	Definition
3D	3 Dimensional
AACE	American Association of Cost Engineers
AAS	atomic absorption spectroscopy
ABS	Australian Bureau of Statistics
ACH Act 2003	*Aboriginal and Cultural Heritage Act* 2003
Ag	Silver
Ai	Abrasion Index
ALS	ALS Minerals Laboratories
Altona	Altona Mining Limited
AR	Abrasion resistant
ARD	acid rock drainage
ARI	average recurrent intervals
AU$	Australian Dollars
Ausenco	Ausenco Engineering Canada Inc.
BMWi	Bond ball mill work index
BRWi	Bond rod mill work index
C$	Canadian dollar
CAPEX	capital expenditure
CCTV	Closed-circuit television
Cementation	Cementation Canada Inc.
CIM	Canadian Institute of Mining, Metallurgy, and Petroleum
CITI SMCC	CITIC SMCC Process Technology Pty. Ltd.
CITIC	China International Trust Investment Corporation
CM	construction management
CMMC, or the Company	Copper Mountain Mining Corporation
CMMPL	Copper Mountain Mining Pty. Ltd. (formerly Altona)
CPI	Consumer Price Index
CRAE	CRA Exploration
CSA	Canadian Standards Association
CSAMT	Controlled Source Audio-Frequency Magneto Telluric
CTC	Cabbage Tree Creek
D&B	Drill and blast
DCS	Distributed control system
DDH	Diamond drill hole
DEHP	Department of Environment and Heritage Protection

Acronyms	***Definition***
DES	Department of Environment and Science
DFS	Definitive Feasibility Study
DGPS	Differential Global Positioning System
DIDO	Drive-in/drive-out
DNRM	Department of Natural Resources and Mines
DNRME	Department of Natural Resources, Mines and Energy
DNRME	Department of Mines and Natural Resources and Energy
DTMR	Department of Transportation and Main Roads
DWi	Drop Weight index
EC	Electrical Conductivity
EIS	Environmental Impact Statement
EM	Electromagnetic
EMP	Environmental Management Plan
EP	Engineering and Procurement
EP Act 1994	*Environmental Protection Act 1994* (Queensland)
EPBC Act 1999	Environment Protection and Biodiversity Conservation Act 1999
EPCM	Engineering, Procurement, and Construction Management
EPM	Exploration Permit for Minerals
EPP	Environmental Protection Policy
ERC	Estimated Rehabilitation Cost
FCA	Freight carrier (or free carrier)
FCA	Free Carrier Agreement
FEL	Front-end-loader
FIFO	Fly-In/Fly-Out
G&A	General and Administrative
George Orr	George Orr and Associates
GPS	Global Positioning System
GST	goods and services tax
Hatch	Hatch Limited
HDPE	High-density polyethylene
HPGR	High pressure grinding rolls
HQ	Drill core size (6.4 cm diameter)
HSE	Health, Safety, and Environmental
HSEP	Health, Safety and Environment Plan
HV	High Voltage
HVAC	Heating, Ventilation, and Air Conditioning
IOCG	Iron-oxide-copper gold
IP	Induced Polarization
IRR	Internal Rate of Return

Acronyms	***Definition***
IT	Information Technology
JORC	Joint Ore Reserves Code—Australasian Code for Reporting of Exploration Results, Mineral Resources, and Ore Reserves
JV	Joint Venture
KCB	Klohn Crippen Berger
KH Morgan	KH Morgan and Associates
Knight Piésold	Knight Piésold Pty. Ltd. (Perth, WA)
LG	Lerchs-Grossmann
LGC	Large Generation Certificate
LOM	Life-of-Mine
MBS	MBS Environmental (Perth, WA)
MERFP Act	*Mineral and Energy Resources (Financial Provisioning) Act 2018*
Merit	Merit Consultants International Inc.
Metso	Metso Corporation
ML	Mining Lease
ML-ARD	Metal Leaching and Acid Rock Drainage
MMG	MMG Limited
MR Act 1989	*Mineral Resources Act 1989* (Queensland)
MRC	Mechanized Raise Climber
MTO	Material Take-offs
NAG	Net/Non-Acid Generation(ing)
NAPCO	North Australian Pastoral Company Pty. Ltd.
NC Act 1992	*Nature Conservation Act 1992* (Queensland)
NCWR 1994	Nature Conservation (Wildlife) Regulation 1994 (Queensland)
NI 43-101	National Instrument for the Standards of Disclosure for Mineral Projects (Canada)
NPV	Net Present Value
NQ	A drill core size (4.8 cm diameter)
NSR	Net Smelter Return
NWPS	North West Power System
OMC	Orway Mineral Consultants
OSA	on-stream analyzer
PAG	Potential Acid-Generating
PanAust	Pan Australian Resources NL
Paterson & Cooke	Paterson & Cooke Consulting Engineers
PAX	Potassium Amyl Xanthate
PCS	Process Control System
PEA	Preliminary Economic Assessment (as defined in NI 43-101)
PEP	Project Execution Plan
pH	Percentage hydrogen

Acronyms	***Definition***
PLC	Programmable Logic Controller
PoO	Plan of Operations
PRC	Progressive Rehabilitation and Closure
QA/QC	Quality Assurance/Quality Control
QEPA	Queensland Environmental Protection Agency
QP	Qualified Person (as defined in NI 43-101)
QR	Queensland Rail
RAB	Rotary Air Blast
RC	Reverse Circulation
RCPL	Roseby Copper Pty Ltd (formerly Bolnisi Logistics)
Rockwater	Rockwater Hydrogeological Consultants
ROM	Run-of-Mine
RQD	Rock Quality Designation
SAG	Semi-Autogenous Grinding
SCL	Stuart Collins Sole Proprietor
SEDAR	System for Electronic Document Analysis and Retrieval
Sedgman	Sedgman Limited
SG	Specific Gravity
SMC	SAG Mill Comminution
SMD	Stirred Mill Detritor
SCP	Stockpile Pond
SS	Scoping Study (as defined in JORC)
STC	Small-Scale Technology Certificates
SunWater	SunWater Limited ....
TC/RC	Treatment Charges and Refining Charges
TDS	Total Dissolved Solids
The Project	Eva Copper Project
TSF	Tailings Storage Facility
UCS	Uniaxial Compressive Strength
UHF	Ultra-High Frequency
Universal	Universal Resources Limited
UPS	Uninterruptible Power Supply
US$	United States Dollars
UTM	Universal Transverse Mercator
VoIP	Voice over Internet Protocol
VTM	Vertimill®
WBS	Work Breakdown Structure
X	X coordinate (E-W)
XRF	X-ray fluorescence

Acronyms	***Definition***
Y	Y coordinate (N-S)
Z	Z coordinate (depth or elevation)

27.2.2 Units of Measure

Acronyms	***Definition***
$	Dollar
%	Percent
¢/kWh	Cents per kilowatt hour
°C	Degrees Celsius
μm	Micrometres (microns)
Au	Gold
cm	Centimetre
Cu	Copper
g	Grams
g/t	Grams per tonne
GL	Gigalitres
h	Hour
h	Hours
h/a	Hours per annum
h/d	Hours per day
ha	Hectare
hp	Horsepower
kg	Kilogram
kL	Kilolitre
km	Kilometre
km/h	Kilometres per hour
km^2	Square kilometres
kPa	Kilopascal
kt	Kilotonne
kV	Kilovolt
kVA	Kilovolt-Ampere
kW	Kilowatt
kWh	Kilowatt hour
L	Litre
L/sec	Litres per second
M	Metre
M	Million

Acronyms	***Definition***
m/h	Metres per hour
m^2	Square metres
m^3	Cubic metres
Ma	Million years (mega annum)
mASL	Metres above sea level (to be interpreted as Australian Hight Datum metres Above Sea Level)
Mg	Magnesium
min	Minutes
mm	Millimetre
mm/a	Millimetres per year
mm/d	Millimetres per day
Mm^3	Million cubic metres
Mn	Manganese
Mo	Molybdenum
Mt	Million tonnes
Mt/a	Million tonnes per year (annum)
MW	Megawatt
No.	Number
oz	Ounces
oz/t	Troy ounces per tonne
Pb	Lead
ppm	Parts per million
sec	Seconds
t/a	Tonnes per year (annum)
t/d	Tonnes per day
t/h	Tonnes per hour
t/m^3	Tonnes per cubic metre
V	Volts
W	Watt
w:o	waste:ore
wt%	Weight Percent (same as w/w)
Zn	Zinc

28 CERTIFICATE OF AUTHORS

28.1 L. Paul Staples, P.Eng.

, L. Paul Staples, P.Eng. of Kaleden, BC, Canada, as an author of this report titled *NI 43-101 Technical Report for the Eva Copper Project Feasibility Study Update, North West Queensland, Australia*, with an effective date of January 31, 2020 (the Technical Report) prepared for Copper Mountain Mining Corporation (CMMC) and dated May 7, 2020, do hereby certify that:

1. I am the Vice President and Global Practice Lead at Ausenco Engineering Canada Inc., with a business address at 855 Homer Street, Vancouver, BC.
2. I am a Metallurgical Engineer by profession with a degree from Queen's University in Kingston ON. I am a registered P.Eng. in the Province of BC, registration #47367.
3. I have practiced my profession for more than 25 years. I have worked in mining projects across Canada, and in South America, Asia Pacific, and Europe.
4. I have read the definition of Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I am responsible for Section 17, and contributed to Sections 1, 3, 18, 25.3, 25.4, 26.2, and 27 of this Technical Report.
6. I have not had prior involvement with the property that is subject to this Technical Report.
7. I am independent of CMMC.
8. I have not visited the Eva Copper Project site.
9. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible, and they have been prepared in compliance with that instrument.
10. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I have read the News Release dated May 7, 2020 and confirm that it is a fair and accurate summary of my sections of this report.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority, and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 7th day of May 2020, at Vancouver, BC, Canada.

Original Signed and Sealed

L. Paul Staples, P.Eng.
Vice President and Global Practice Lead
Ausenco

28.2 Peter Holbek, MSc., PGeo.

I, Peter Holbek, M.Sc., P.Geo., of Vancouver, BC, as an author of this report titled *NI 43-101 Technical Report for the Eva Copper Project Feasibility Study Update, North West Queensland, Australia*, with an effective date of January 31, 2020 (the Technical Report) prepared for Copper Mountain Mining Corporation (CMMC) and dated May 7, 2020, do hereby certify that:

1. I am the Vice President, Exploration, at CMMC, with a business address at Suite 1700, 700 West Pender Street, Vancouver, BC, V6C 1G8, Canada, and have worked with CMMC during the preparation of this report.
2. I am a Licensed Professional Geoscientist by profession (registered by Engineers and Geoscientists, BC) (University of British Columbia, BSc Hons, 1980, MSc in Geology, 1988).
3. I have practiced my profession for more than 35 years. I have worked in exploration and mining projects across Canada, and in South America and Europe.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I am responsible for Sections 7 to 12 inclusive, and Section 14, and contributed to Sections 1, 3, 25, 26, and 27 of the Technical Report.
6. I am not independent of CMMC.
7. I visited the Eva Copper Project site over a 3-day period in February 2017.
8. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible, and they have been prepared in compliance with that instrument.
9. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I have read the News Release dated May 7, 2020 and confirm that it is a fair and accurate summary of my sections of this report.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority, and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 7th day of May 2020, at Vancouver, BC, Canada.

Original Signed and Sealed
Peter Holbek
Copper Mountain Mining Corporation
Vice President Exploration

28.3 Stuart Collins, P.E.

I, Stuart E. Collins, P.E., of Denver, Colorado, USA, as an author of this report titled *NI 43-101 Technical Report for the Eva Copper Project Feasibility Study Update, North West Queensland, Australia*, with an effective date of January 31, 2020 (the Technical Report) prepared for Copper Mountain Mining Corporation (CMMC) and dated May 7, 2020, do hereby certify that:

1. I am an Independent Professional Mining Engineer located at 4500 Cherry Creek South Drive, Suite 1040, Denver, Colorado, USA, 80246.
2. I am a Registered Professional Engineer in the state of Colorado (#29455) (South Dakota School of Mines and Technology – BSc in Mining Engineering, 1985). I have been a member of the Society for Mining, Metallurgy, and Exploration (SME) since 1985, and a Registered Member (#612514) since September 2006.
3. I have worked as a mining engineer for a total of 34 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Conducting reviews and reports as a consultant on numerous explorations, development, and production mining projects around the world for due diligence and regulatory requirements.
    - Performing mine engineering, mine management, mine operations, and mine financial analyses involving copper, gold, silver, nickel, cobalt, uranium, iron ore, chromium, coal, and other base metals deposits located in Australia, Papua New Guinea, USA, Canada, Mexico, Turkey, Bolivia, Chile, Brazil, Costa Rica, Peru, Argentina, and Colombia.
    - Engineering Manager for several mining-related companies.
    - Business Development for a small, privately-owned mining company in Colorado.
    - Operations supervisor at a large gold mine in Nevada, USA.
    - Involvement with the development and operation of a small underground gold mine in Arizona, USA.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I am responsible for Sections 15, 16 and 21 (mining CAPEX and OPEX), and contributed to Sections 1, 3, 22, 25, 26, and 27 of the Technical Report.
6. I have had prior involvement with the property that is the subject of this Technical Report.
7. I am independent of CMMC.
8. I visited the Eva Copper Project from 19 to 23 September 2018.
9. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible, and they have been prepared in compliance with that instrument.
10. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I have read the News Release dated May 7, 2020, and confirm that this news release is a fair and accurate summary of my sections of this report.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority, and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 7th day of May 2020, at Denver, Colorado, USA.

Original Signed and Sealed

Stuart E. Collins, P.E.

28.4 Mike Westendorf, P.Eng.

I, Mike Westendorf, P.Eng., of Vancouver, BC, as an author of this report titled *NI 43-101 Technical Report for the Eva Copper Project Feasibility Study Update, North West Queensland, Australia*, with an effective date of January 31, 2020 (the Technical Report) prepared for Copper Mountain Mining Corporation (CMMC) and dated May 7, 2020, do hereby certify that:

1. I am the Director of Metallurgy with CMMC with a business address at Suite 1700, 700 West Pender Street, Vancouver, BC, Canada, and have worked with CMMC during the preparation of this report.
2. I am a Metallurgical Engineer by profession (University British Columbia), registered with Engineers and Geoscientists British Columbia (EGBC).
3. I have practiced my profession for more than 12 years. I have been directly involved in mining and mineral processing projects in Canada.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I am responsible for Sections 13 and 21 (process plant and Infrastructure OPEX), and contributed to Sections 1, 3, 25, 26, and 27 of this Technical Report.
6. I have had prior involvement with the property that is the subject of this Technical Report.
7. I am not independent of CMMC.
8. I have visited the Eva Copper Project site from July 22 to 25, 2019.
9. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible, and they have been prepared in compliance with that instrument.
10. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I have read the News Release dated May 7, 2020 and confirm that it is a fair and accurate summary of my sections of this report.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority, and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 7th day of May 2020, at Vancouver, BC, Canada.

Original Signed and Sealed

Mike Westendorf, P.Eng.
Director of Metallurgy
Copper Mountain Mining Corporation

28.5 Alistair Kent, ME, P.Eng.

I, Alistair Kent of Vancouver, BC, Canada, as an author of this report titled *NI 43-101 NI 43-101 Technical Report for the Eva Copper Project Feasibility Study Update, North West Queensland, Australia*, with an effective date of January 31, 2020 (the Technical Report) prepared for Copper Mountain Mining Corporation (CMMC) and dated May 7, 2020, do hereby certify that:

1. I am a Senior Project Manager with Merit Consultants International, Vancouver, BC, Canada, and have worked with CMMC during the preparation of this report.
2. I am a Civil Engineer by profession (University of Auckland – ME in Civil Engineering, 1977), and I am a member in good standing of Engineers and Geoscientists BC (License #13842).
3. I have 40 years of relevant experience in engineering and planning mining projects, including Oracle Copper in Arizona, Victoria Gold, Copper North, and Selwyn Chihong zinc in YT, Highland Valley Copper in BC, and Meadowbank gold and High Lake gold/copper in NU.
4. I have read the definition of Qualified Person set out in the National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I am responsible for Sections 21.1, 21.3, and 21.4 (Initial Capital Cost, excluding mining) and 24, and contributed to Sections 1, 3, 25, and 27 of the Technical Report.
6. I am independent of CMMC.
7. I have not visited the Eva Copper Project site.
8. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible, and they have been prepared in compliance with that instrument.
9. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I have read the News Release dated May 7, 2020 and confirm that it is a fair and accurate summary of my sections of this report.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority, and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this Dated this 7th day of May 2020, at Vancouver, BC, Canada.

Original Signed and Sealed

Alistair Kent
Senior Project Manager
Merit Consultants International

28.6 David Johns, MIEAust, CPEng., RPEQ, Pr Eng.

I, David Johns, of Brisbane, Queensland, Australia, as an author of this report titled *NI 43-101 Technical Report for the Eva Copper Project Feasibility Study Update, North West Queensland, Australia*, with an effective date of January 31, 2020 (the Technical Report) prepared for Copper Mountain Mining Corporation (CMMC) and dated May 7, 2020, do hereby certify that:

1. I am a Senior Geotechnical Engineer and Associate with Klohn Crippen Berger Australia (KCB), with a business address at Level 1, 154 Melbourne Street, Brisbane, QLD, Australia, and have worked with CMMC during the preparation of this report.
2. I am a Civil and Geotechnical Engineer by profession (University of the Witwatersrand, BSc (Eng) and MSc (Eng)), a Registered Professional Engineer of Queensland (RPEQ) registered with the Board of Professional Engineers of Queensland, a Chartered Engineer registered with the Institute of Engineers Australia (MIEAust), and a Professional Engineer (Pr Eng) registered with the Engineering Council of South Africa.
3. I have continuously practiced my profession for more than 15 years. I have been directly involved in mining and mineral processing projects in Australia, Namibia, Botswana, South Africa, Democratic Republic of Congo, and Papua New Guinea.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I am responsible for sub-sections 18.4 and 18.5, and contributed to Sections 1, 3, 25, 26, and 27 of the Technical Report.
6. I have had no prior involvement with the property that is subject to the Technical Report.
7. I am independent of CMMC.
8. I have not visited the Eva Copper Project site.
9. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible, and they have been prepared in compliance with that instrument.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I have read the News Release dated May 7, 2020 and confirm that this news release is a fair and accurate summary of my sections of this report.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority, and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 7th day of May 2020, at Brisbane, QLD, Australia.

Original Signed

David Johns, MIEAust, CPEng, RPEQ, Pr.Eng
Associate, Senior Geotechnical Engineer
Klohn Crippen Berger Australia

28.7 Roland Bartsch, MSc, BSc. (Hons), MAusIMM, MAIG

I, Roland Bartsch, of Perth, Western Australia, as an author of this report titled *NI 43-101 Technical Report for the Eva Copper Project Feasibility Study Update, North West Queensland, Australia*, with an effective date of January 31, 2020 (the Technical Report) prepared for Copper Mountain Mining Corporation (CMMC) and dated May 7, 2020, do hereby certify that:

1. I am a Vice President and Country Manager Australia with Copper Mountain Mining Pty. Ltd. of Perth, WA, Australia, a wholly-owned subsidiary of Copper Mountain Mining Corporation, and have worked with CMMPL during the preparation of this report.
2. I am a Geologist by profession (University of British Columbia – MSc in Geology; University of New England, BSc Hons in Geology). I am registered with the Australasian Institute of Mining and Metallurgy as a Member (MAusIMM), and with the Australian Institute of Geologists as a Member (MAIG).
3. I have practiced my profession for more than 30 years. I have been directly involved in or provided consulting services on exploration, mining, and mineral processing projects in Australia, Indonesia, Solomon Islands, Greenland, Canada, USA, Mexico, Chile, Peru, Burkina Faso, Mauritania, Brazil, South Africa, and Sweden.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101), and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
5. I am responsible for the Sections 4 to 6 inclusive, 20, and 23, and contributed to Sections 1, 3, 25, 26, and 27 of the Technical Report.
6. I have had prior involvement with the property that is subject to the Technical Report. I was the General Manager Exploration for Altona Mining Limited (acquired by CMMC and renamed CMMPL) and a primary author of preceding Feasibility Studies by Altona and CMMC.
7. I am not independent of Copper Mountain Mining Corporation.
8. I have visited the Eva Copper Project site on numerous occasions.
9. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible, and they have been prepared in compliance with that instrument.
10. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I have read the News Release dated May 7, 2020 and confirm that this news release is a fair and accurate summary of my sections of this report.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority, and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 7th day of May 2020, at Vancouver, BC, Canada.

Original Signed and Sealed

Roland Bartsch
VP and Country Manager – Australia
Copper Mountain Mining Pty. Ltd.

28.8 Richard Klue, NHD Ext. Met, BCom, FSAIMM

I, Edward Richard Klue, of Vancouver, BC, Canada, as an author of this report titled *NI 43-101 Technical Report for the Eva Copper Project Feasibility Study Update, North West Queensland, Australia*, with an effective date of January 31, 2020 (the Technical Report) prepared for Copper Mountain Mining Corporation (CMMC) and dated May 7, 2020, do hereby certify that:

1. I am a Vice President with Copper Mountain Mining Corporation with a business address at Suite 1700, 700 West Pender Street, Vancouver, BC, Canada, and have worked with CMMC during the preparation of this report.
2. I am a Metallurgical Engineer by profession (University of Johannesburg – NHD Ext. Met.), (University of South Africa (UNISA) – B.Com) registered with the South African Institute of Mining and Metallurgy as a Fellow (FSAIMM).
3. I have continuously practiced my profession for more than 35 years. I have been directly involved in mining and mineral processing projects in Canada, USA, Mexico, Chile, Cuba, Australia, Namibia, Laos, India, South Africa, the Democratic Republic of Congo, Zambia, Cameroon, Iran, Russia, the United Kingdom, Norway, Spain, and Ireland.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of NI 43-101.
5. This certificate applies to the Technical Report with an effective date of January 31, 2020.
6. I am responsible for Sections 2, 3, 18, 19, and 22, and contributed to Sections 1, 25, 26, and 27 of the Technical Report.
7. I have had prior involvement with the property that is subject to the Technical Report.
8. I am not independent of CMMC.
9. I visited the Eva Copper Project site from 15 to 17 October 2018.
10. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible, and they have been prepared in compliance with that instrument.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
12. I have read the News Release dated May 7, 2020 and confirm that this news release is a fair and accurate summary of my sections of this report.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority, and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 7th day of May 2020, at Vancouver, BC, Canada.

Original Signed and Sealed

(Edward) Richard Klue
Vice President Technical Services
Copper Mountain Mining Corporation